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PREFACE

In this volume are the papers and discussions on Mining, Milling, Geology and Coal that were presented at the Lake Superior meeting, August, 1920, the New York meeting, February, 1921, and the Wilkes-Barre meeting, September, 1921.

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Year of Election to Honorary Membership	Name	Residence
1876.	Prof. Richard Akerman	Stockholm, Sweden.
1913.	Dr. Frank Dawson Adams	Montreal, Canada.
1921.	William C. Blackett	London, England.
1920.	Henry Sturgis Drinker	Bethlehem, Pa.
1921.	Ferdinand Foch	Paris, France.
1888.	Prof. Haton de la Goupilliere	Paris, France.
1921.	F. W. Harbord	London, England.
1906.	Sir Robert A. Hadfield	London, England.
1921.	H. O. Hofman	Cambridge, Mass.
1917.	Herbert Hoover	Stanford University, Cal.
1919.	Robert W. Hunt	Chicago, Ill.
1915.	James Furman Kemp	New York, N. Y.
1905.	Prof. Henri Louis Le Chatelier	Paris, France.
1921.	C. McDermid	London, England.
1913.	Ezequiel Ordoñez	Mexico City, Mexico.
1909.	Alexandre Pourcel	Paris, France.
1911.	Robert H. Richards	Boston, Mass.
1906.	John E. Stead	Middlesbrough, England.
1907.	Charles D. Walcott	Washington, D. C.

Honorary Members (*Deceased*)

	Name	Year of Decease
1872.	Sir Lowthian Bell	1904
1905.	Andrew Carnegie	1919
1892.	A. Del Castillo	1895
1902.	Manuel Maria Contreras	1902
1888.	A. Daubrée	1896
1906.	James Douglas	1918
1884.	Thomas M. Drown	1904
1890.	Moritz Gaetzschmann	1895
1873.	L. Gruner	1883
1891.	Bruno Kerl	1905
1895.	Joseph Le Conte	1901
1891.	J. P. Lesley	1903
1899.	Floris Osmond	1912
1890.	Adolph Patera	1890
1886.	John Percy	1889
1888.	Franz Posepny	1895
1911.	Rossiter W. Raymond	1918
1884.	Theodor Richter	1898
1899.	W. C. Roberts-Austen	1902
1890.	Albert Serlo	1898
1880.	C. William Siemens	1883
1909.	James M. Swank	1914
1902.	Prof. Dimitry Constantin Tschernoff	1921
1872.	David Thomas	1882
1873.	Peter R. von Tunner	1897
1885.	Hermann Wedding	1908
1910.	Prof. Tsunashiro Wada	1920

Meeting in the Lake Superior District

THE 122d meeting of the Institute was held in the Lake Superior Copper and Iron Country Aug. 20 to Sept. 3, 1920, with an approximate registration of 1100 members and guests. This was the first visit of the Institute to this vicinity since 1904, although previously it held meetings there every six or seven years.

At Houghton, the Houghton Club was the headquarters and was thrown open to members. On Monday evening, a dance was given at the Onigaming Yacht Club and an overflow dance at the Michigan College of Mines. On Tuesday morning the party, largely augmented by train arrivals, left in autos for the different inspection trips at the mines and reduction works, returning to Houghton for dinner. A technical session was held at the Michigan School of Mines, presided over by Dr. F. W. McNair, when the following papers were presented.

Handling and Treatment of Rock-drill Steel at the Copper Range Mines. By H. T. Mercer and A. C. Paulsen.

Steam Regenerators Reduce Coal Consumption. By W. H. Schacht.

Athens System of Mining. By S. R. Elliott.

Mechanical Ventilation at the Lake Mine. By Lucien Eaton.

Building Reinforced-concrete Shaft Houses. By J. Ellzey Hayden and Lucien Eaton.

Special trains during the night took some of the party to Marquette and the others to Iron Mountain.

The party visiting the Marquette Range spent the morning inspecting the Pioneer Furnace and chemical plant of the Cleveland Cliffs Iron Co., leaving just in time to reach the Wawonowin Golf Club for luncheon. During the afternoon the party visited the Athens and Negaunee mines.

The Menominee party in the morning visited several shafts of the Chapin mine and the hydro-electric plants of this vicinity; in the afternoon, the Brier Hill mine and various shafts of the Vulcan mine of the Midvale Steel Co. Luncheon was served at the Pine Grove Country Club at Iron Mountain.

In the evening the party reassembled at Iron Mountain, where films showing Midvale Steel operations were shown and explained by a steel company representative. After the films, Mr. Hoover told of the part taken by his associate engineers in the Belgian Relief, the Food Administration, and the Supreme Economic Council. The special train then left for Minneapolis.

During the forenoon of Aug. 26, the following technical sessions were held at the University of Minnesota:

Session on Mine Practice

Herbert Hoover presiding

Modern Commercial Explosives and Their Uses. By A. J. Strane.

Standardizing by the North Butte Mining Co. By Robert Linton.

Mining Methods and Costs at the United Verde Mine. By H. DeWitt Smith and W. H. Sirdevan.

Hoisting Equipment at Utah-Apex Mine. By Joseph A. Norden and A. R. Willson.

Development Practice in the Wisconsin Zinc District. By Edwin R. Shorey.

A Reflecting Microscope for the Mining Engineer. By W. Myron Davy.

Care of Rock Drills. By Howard R. Drullard.

Session on Geology and Resources

W. H. Emmons presiding

The Dip Needle in Stratigraphy. By H. R. Aldrich.

Exploration Methods on the Gogebic Range. By W. O. Hotchkiss.

Application of Ball-mills in Southeast Missouri. By Lewis A. Delano and Harold Rabling.

Geology and Ore Deposits of the Jerome District. By Louis E. Reber, Jr.

The entire party was taken to St. Paul in automobiles for luncheon at the St. Paul hotel, where short speeches were made by Mr. Hoover, Senator Elliott and others, and the afternoon was spent in sightseeing.

The banquet in the evening at the Curtis Hotel was an unusual success. After the first course, the lights went out and a party of miners in oil skins with miners' lamps on their hats entered the room; after singing a couple of much appreciated songs, they produced several bags of high-grade samples and distributed as souvenirs of the iron country cast-iron ash trays of artistic design, which Prof. A. L. Walker pronounced almost equal to the famous Kishtim castings. The speeches were brief because the train for the iron ranges left at 11.30 o'clock.

William C. Edgar, the toastmaster, was introduced by Philip N. Moore. Fred B. Snyder, president of the Board of Regents of the University, spoke of the resources of Minnesota and the tax problems facing the iron industry. Horace V. Winchell was unfortunately in Alaska, having been called by telegram unexpectedly to make an examination before navigation closed. He sent a telegram of regret, with a message of congratulation and esteem for Herbert Hoover. At the suggestion of John E. Hodge, acting chairman of the Committee on Arrangements, a telegram was sent to Winchell, addressed, because he is so well known, only H. V. Winchell, Alaska; this was approved by a rising vote. J. E. Spurr recounted some of his early experiences in the iron ranges, when Hibbing had only one hotel with two rooms, one upstairs and one down.

Herbert Hoover was then introduced amid great applause. His speech was printed in MINING AND METALLURGY, October, 1920.

Friday morning, the party awoke on the Mesabi Range at Eveleth, where they were the guests of the Engineers Club of Northern Minnesota. After an inspection of the Leonidas mine, washing and screening plant, the party traveled, by automobile, to Virginia where it visited the Virginia & Rainy Lake lumber mill, the largest in the world, and the iron mines. Luncheon was served on the special train. In the afternoon, the party visited the mines at Chisholm and Hibbing, and was taken on two trains of flat cars through the Hull-Rust-Mahoning, the largest open-pit iron mine in the world.

About one-third of the party went, by automobile, to Babbitt to view the new Hayden-Stone-Jackling development where a 4000-ton plant to beneficiate the low-grade taconite is being built.

A picnic supper was served at Bennett Park. The ladies then went to the Hibbing Public Library, where they were entertained by the ladies of Hibbing, and the men attended a technical session at the Lincoln High School. W. G. Swart presided and the following papers were presented and discussed:

Utilization of Titaniferous Iron Ore. By J. A. Heskett.
Casting and Molding Steel Ingots. By Emil Gathmann.
The Acid Bessemer Process. By Richard S. McCaffery.
A New Occurrence of Pro-Eutectoid Ferrite. By Charles Y. Clayton.

After the technical session the men went to the armory to see several boxing bouts.

During the night the special train was moved to Coleraine where the mine was visited, and the Trout Lake washing plant, which has a capacity of 25,000 tons of concentrates in 10 hours, was inspected. At 10 o'clock the train left for Duluth, where Saturday afternoon was spent visiting the ore docks, steel plant, shipbuilding works, and sight-seeing; and at 5 o'clock the party boarded the steamer "Rotarian" for a trip around the harbor and a "close-up" of the ore-loading and coal-unloading facilities. A box supper was served on the boat.

Woman's Auxiliary at the Lake Superior Meeting

The Woman's Auxiliary to the A. I. M. E. was active and participated in many functions on the Institute trip to the Lake Superior country and visited many mines and plants.

Monday night the ladies participated in the informal reception to Mr. Hoover and the dance at the Onigaming Yacht Club. Tuesday night the ladies of Houghton and neighboring copper towns téndered a reception to Mrs. Dwight in the auditorium of the School of Mines. Mrs. Dwight spoke briefly of the objects and work of the Auxiliary and urged the formation of a Houghton Section. Mrs. Percy E. Barbour spoke of the luncheons held monthly by the New York Section and the

benefit derived from this frequent intercourse. This was followed by a dance in the gymnasium, which broke up only in time to catch the special train for the iron ranges.

Wednesday, at Vulcan, Mrs. William Kelly entertained the Woman's Auxiliary at tea in her home. Surrounded by flowers and trees, this was one of the most beautiful spots of the trip.

In Minneapolis, on Thursday, the Auxiliary met at the home of Mrs. Horace V. Winchell. Mrs. Dwight spoke again on the work of the Auxiliary, urged a membership campaign and formation of a local section. Mrs. George D. Barron spoke on the scholarships founded by the Auxiliary and explained their purpose and present standing.

On Friday evening, the ladies of Hibbing held open house for the ladies of the party and residents at the Library. A very interesting program of entertainment was enjoyably rendered and light refreshments were served. It was a delightful evening and the hospitality of the ladies of the Range was much appreciated.

The One Hundred and Twenty-third Meeting of the Institute

THE 123d meeting of the Institute was held in New York, Feb. 14 to 17, 1921. The total registration was 1199, as compared with 1138 at the New York meeting in 1920. The weather was a strange and welcome contrast to that of the previous year, when New York was buried under 6 ft. of snow as the result of the fiercest blizzard in its history.

A complete program of the Technical Sessions begins on p. xv. The showings of moving picture films, at two sessions, were largely attended. Those of the United States Steel Corpn. welfare work were under the auspices of the Committee on Industrial Relations and were shown without an accompanying speaker. The film on the Present Day Production of Sulfur (taken in coöperation with the Texas Gulf Sulphur Co.) was accompanied by a paper presented in person by Col. Raymond F. Bacon and gave in interesting detail the work of the Gulf Sulphur Co. in Texas. The other two films shown were on the Mining and Production of Asbestos (taken in coöperation with the H. W. Johns-Manville Co.) and the Making of Steel (with the coöperation of the American Rolling Mill Co.), all of which were made under the auspices of the United States Bureau of Mines.

Each noon luncheons were served on the fifth floor, in which the ladies participated. Larger numbers were served than ever before.

The smoker was held for the first time in the auditorium and was attended by about 700. The program was furnished by professional entertainers and by E. P. Mathewson, chairman, and concluded with several reels of moving pictures.

On Tuesday evening, a professional entertainment was provided in the auditorium for the members of the Institute and the members of the Woman's Auxiliary and their friends, followed later by refreshments and dancing on the fifth floor.

Tuesday afternoon, the Institute members were again treated to the pleasure of visiting Senator W. A. Clark's magnificent home and art galleries. During the view, there was an organ recital and later tea was served.

On Wednesday afternoon, the Woman's Auxiliary entertained at a fashion show at Lucile's. A few of the engineers accompanied the ladies to this function, which proved to be a most fascinating affair. Tea was served afterwards at the tea room of the American Committee for Devastated France, on 39th Street.

On Thursday, instead of visiting some metallurgical or manufacturing establishment, about 200 members and guests were taken through the

museums at 156th Street and Broadway. A private view of the Indian collections in the Museum of the American Indian, Heye Foundation, much in advance of opening to the public, was afforded the members. This museum is devoted entirely to the American Indian. It had its inception 20 years ago and is now housed in a beautiful building facing the court which is surrounded by the other museums to be mentioned. The sole aim is to gather and preserve for students everything useful in illustrating the life and history of the aborigines of the Western hemisphere. This was followed by a visit to the Hispanic Society of America which, through the munificence of Mr. Huntington, has the finest collection of Hispanic art and literature in the Western hemisphere. At the American Numismatic Society, many interesting coins and medals were exhibited as well as a most interesting collection of bronzes by Mrs. Sheridan. Those that attracted the most attention were the recently completed ones of Lenine and Trotzky which, perhaps more than anything else, despite the excellence of her work, have brought her such quick fame.

After visiting the American Geographical Society, the party went to the May Flower Restaurant, at the corner of 157th Street, for lunch. Following the luncheon, the party visited the Hall of Fame at New York University and then the ladies hurried to the Cosmopolitan Club, where they were entertained at tea by Mrs. Hoover.

The Annual Banquet

The annual banquet, held at the Waldorf, had a larger attendance than any previous one. The dinner, which was preceded by a reception to the outgoing and incoming presidents and their wives, was a brilliant affair. Balloons and toy airships at each cover were soon soaring through the air. The favors were monel-metal watch chains, with gold snap and swivel, presented by the International Nickel Co.

The following description of the function is taken from the New York *Evening World*, whose cartoonist successfully disguised some of our best known members:

An engineered banquet—scientific in its perfection and perfect in its science, a thing to delight equally a poet and a machinist—was given at the Waldorf last night by the American Institute of Mining and Metallurgical Engineers, who deserve the additional title of Dining and Dancing Engineers.

Even the speechmaking was controlled as cleverly as the mainspring of a Swiss watch, the third man on the program being introduced firmly for the purpose of being looked at and not listened to; and so absolute was the sway of the chief engineer that this third man did not venture to say "I thank you."

It was a model. Even the time that the ladies would require to dress had been foretold as accurately as an astronomer foretells an eclipse. And the final triumph of the evening came when the toastmaster was able to tell all present that if they wanted to go home and take a nap and return at 2 A.M. for the dancing, the dancing would be ready for them.

That toastmaster was William Lawrence Saunders of Plainfield, N. J., a poet, a prophet, an engineer and an inventor. Away back in the seventies or eighties he invented an apparatus for sub-aqueous drilling, although Prohibition had not even been thought of at that time, and there was no apparent reason why anybody should drill sub-aqua. The drill was not on hand last night and everybody drank water.

But even this was planned. The program boasted that "the society is fifty years old this year and still going strong" and against it was set: "The banquet will be dry."

Herbert Hoover spoke, and in less than sixty seconds he had said the thing that he chiefly wanted to say—that 28 states had gone "over the top" in the $33,000,000 drive for European orphans, and that the whole sum certainly will be subscribed by the end of the month. The rest of what he said was a brief report of what the society had accomplished since he took the presidency of it a year ago. He yielded the presidency last night to Edwin Ludlow.

The man who was introduced to be looked at only, not listened to, was Henry S. Drinker of Bethlehem, Pa. The Institute was founded in 1871 and Mr. Drinker was one of the founders. He is one of the three still living, the others being Edmund C. Pechin of Buchanan, Va., and Willard P. Ward of New York.

Technical Sessions

Mining and Milling

R. M. Catlin presiding

Calculation of Ore Tonnage and Grade from Drill-hole Samples. By James E. Harding.

Rate of Formation of Copper Sulfate Stalactites. By Graham John Mitchell.

Relation of Air Pressure to Drilling Speed of Hammer Drills. By H. W. Seamon.

Time Factor in Depletion of Mines. By John W. Roberts.

Non-ferrous Metallurgy

E. P. Mathewson presiding

Steel Chimneys and Their Linings in Copper Smelting Plants. By A. G. McGregor.

Roasting and Chloridizing Bolivian Silver-tin Ores. By M. G. F. Sohnlein.

Treatment Tests on Ores of Consolidated Coppermines Co. By Robert Linton.

Tooele Flue-type Cottrell Treater. By A. B. Young.

Electrolytic Zinc Plant of the Anaconda Copper Mining Co., at Great Falls, Mont. By Frederick Laist, F. F. Frick, J. O. Elton and R. B. Caples.

Non-metallic Minerals

H. Ries presiding

* Potash Deposits of Alsace. By Hoyt S. Gale.
Important Factors in Talc Milling Efficiency. By Raymond B. Ladoo.
Judging the Quality of Portland Cement. By R. J. Colony.
By-product Expansion in the Non-metallic Mineral Industries. By Oliver Bowles.

Special Addresses

The Rehabilitation of Devastated French Coal Mines. By George S. Rice.
Geology of Pachuca and El Oro. By H. V. Winchell.

Iron and Steel

Jos. W. Richards presiding

Measurement of Blast-furnace Gas. By D. L. Ward and R. S. Reed.
Importance of Hardness of Blast-furnace Coke. By Owen R. Rice.
Manufacture of Ferromanganese in the Electric Furnace. By Robert M. Keeney and Jay Lonergan.
Electric Furnace in the Iron Foundry. By Richard Moldenke.

Institute of Metals Division

Wm. H. Bassett presiding

Effect of Temperature, Deformation, Grain Size, and Rate of Loading on the Mechanical Properties of Metals. By W. P. Sykes.
Artillery Cartridge Cases. By J. Burns Read and S. Tour.

Coal

R. V. Norris presiding

Alaskan Coal Fields. By George Watkin Evans.
Mine-water Neutralizing Plant at Calumet Mine. By L. D. Tracy.
Run-off and Mine Drainage. By Howard N. Eavenson.
Skip Hoisting for Coal Mines. By Andrews Allen and J. A. Garcia.
Coal Pillar Drawing Methods in Europe. By George S. Rice.
Report and Suggestions of Committee on Standard System of Accounting and Analysis of Cost of Production, National Coal Association.
Pillar Drawing in Thick Coal Seams. By G. B. Pryde and R. M. Magraw.

Iron and Steel

A. A. Stevenson presiding

Nitrogen in Steel and Erosion of Guns. By H. E. Wheeler.
Static and Dynamic Tension Tests on Nickel Steel. By J. J Thomas and J. H. Nead.
Surface Changes of Carbon Steels Heated in Vacuo. By E. H. Hemingway and G. R. Ensminger.
Chemical Equilibrium between Iron, Carbon and Oxygen. By A. Matsubara.
Molybdenum Steels. By John A. Mathews.

Petroleum and Gas

Ralph Arnold presiding

Principles Governing Mexican Taxation of Petroleum. By V. R. Garfias.

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* Not published by the Institute. *Bull.* 715 B., U. S. Geol. Survey.

Modified Oil-well Depletion Curves. By Arthur Knapp.

Barrel-day Values. Glenn H. Alvey, and Alden W. Foster.

Secondary Intrusive Origin of Gulf Coastal Plain Salt Domes. By W. G. Matteson.

Isostatic Adjustments on a Minor Scale in Their Relation to Oil Domes. By M. Albertson.

Carbon Ratios of Coals in the West Virginia Oil Fields. By David B. Reger.

* Statistical Review of the Mid-Continent Field, 1912–1921 Inclusive. By Mowry Bates and Bernard H. Lasky.

Presentation of Annual Reports, Industrial Relations Sub-committees

T. T. Read presiding

Americanization. By Charles R. Hook.

Cripples in Industry. By H. N. Wolflin.

Employment. By Sidney Rolle.

Mental Factors in Industry. By T T. Read.

Prevention of Illness. By A. J. Lanza.

Housing. By Will. L. Clark.

Breakage and Heat Treatment of Drill Steel

Benj. F. Tillson presiding

Discussion of a proposed research on the Breakage and Heat Treatment of Drill Steel, in coöperation with the United States Bureau of Mines, American Society for Steel Treating, and Engineering Division of National Research Council.

Industrial Relations

Robert Linton presiding

The Health of the Underground Worker. By A. J. Lanza.

Dust-ventilation Studies in Metal Mines. By D. Harrington.

A Symposium on Employment Problems. Introduced by J. N. Houser: Service Dept. of the Tennessee Copper Co. and S. R. Rectanus: Hiring and Placing of Men.

Woman's Auxiliary

The fifth annual meeting of the Woman's Auxiliary took place at Institute headquarters on Feb. 16, 1921, the president, Mrs. Arthur S. Dwight, presiding. Minutes of the last annual meeting were read by the secretary, Mrs. Reynders. The treasurer's report showed that the receipts during the year were $606.45 and the disbursements $382.63, which, with cash on hand on Feb. 1, 1920, leaves a balance on hand of $437.33.

The Educational Fund Committee reported that James B. Christie, nominated by the Anaconda Section, had been awarded a scholarship at the University of California. The committee also reported that the receipts for the year, including interest and deposits, amounted to $1417.30 and the disbursements, chiefly for printing, amounted to $44.31,

* Not published by the Institute.

leaving a balance of $1372.99 which includes the $214.39 invested in Liberty Bonds.

Reports were read from the various local sections of the Auxiliary. Mrs. Hoover was appointed chairman of the general committee on membership and Mrs. Westervelt was made chairman of the committee to revise the Constitution of the Auxiliary.

The meeting closed with an address by Professor James Kemp, of Columbia University. Professor Kemp gave a comprehensive utline of the history of mining, in particular as the industry in later years developed a system of education for the mining engineer. He showed how gradually but persistently progress was marked in those countries where miners possessed a knowledge of mathematics, chemistry and geology, and mining schools increased in number and breadth until now it is generally acknowledged that even four years of scientific training leaves much room for further research and specialization. A full account of the Woman's Auxiliary meeting was published in MINING AND METALLURGY, March, 1921.

Semi-centennial Meeting at Wilkes-Barre

The meeting of the A. I. M. E. at Wilkes-Barre, Pa., Sept. 12 to 15, 1921, celebrated the fiftieth anniversary of the Institute. It was at Wilkes-Barre, in 1871, that the foundation was laid for the great institution that now includes in its roster the greatest names in mining engineering in every civilized country of the world. The little coterie of 1871 has grown to more than 10,000 members in 1921.

The headquarters of the convention was the Irem Temple, a building well adapted for the purpose.

Monday morning was given over to registration and general personal meetings and conferences. In the afternoon a technical session was held, beginning at 2:30 p. m. This session was devoted to general papers on coal, and was presided over by W. J. Richards, president of the Philadelphia & Reading Coal & Iron Co. The following papers were presented:

Mechanical Mining of Anthracite. By H. D. Kynor.
The Ashley Planes. By C. H. Stein.
Anthracoal. By Donald Markle.
The Slush Problem. By John Griffen.

In the evening, two technical sessions were held:

Coal Session

C. F. Huber presiding

The Lynch Plant. By Howard N. Eavenson.
General Description Anthracite Field, with maps and sections. By E. W. Parker.

Metal Section

J. V. W. Reynders presiding

Application in Rolling of Effects of Carbon, Phosphorus and Manganese on Mechanical Properties of Steel. By W. R. Webster.

Thacher Process for Molding and Casting Propeller Blades and Wheels. By E. Touceda.

Making a 5 per cent. Nickel-cast-iron Alloy in an Electric Furnace. By D. N. Witman.

On Tuesday, the day's events began with an automobile trip through the Wyoming Valley. Automobiles were provided for the entire party, which left Irem Temple about nine o'clock. The printed program of the meeting, an excellently prepared booklet of 76 pages, provided each member with a catalog of the interesting points of the trip. The first stop was at the Marvine Colliery plant of the Hudson Coal Co., where the visiting engineers were invited to inspect the installation. The party then proceeded to the building of the International Correspondence School, where guides explained the purposes and accomplishments of

the institution. Subsequently a luncheon was served by the school management.

A technical session was held in the afternoon, the International Correspondence Schools placing their auditorium at the disposal of the Institute. At this session, which was in charge of William Griffith, a paper was read by Douglas Bunting on "Mine Fires," in which the occurrence and extinguishment of specific fires in the district was described. The discussion on this paper was so long that none of the other papers on the program could be presented.

A session on Americanization was held at Scranton, at which Arthur Notman presided. The papers presented were:

Americanization in Mining Communities of Pennsylvania. By E. E. Bach.
Americanization in the Anthracite Field. By F. E. Zerbey.
Americanization—What is It? By S. E. Weber.
Americanization Work of the Y. M. C. A. Among Industrial Workers. By Peter Roberts.
Observations on Americanization. By J. A. Water.
Americanization in the Butte District. By L. D. Frink.

At the close of these sessions, the party returned to Wilkes-Barre by automobile. In the evening a technical session was held in Irem Temple, R. A. Quin presiding. The papers presented were:

Power Installation at Coverdale. By C. M. Means.
Octagonal Ventilation Shaft of Davis-Daly Copper Co. By J. L. Bruce.
Application of Pulverized Coal to Boilers. By J. W. Fuller.
Electric Power a Factor in the Anthracite Field. By W. A. Thomas.
Determination of Electrical Equipment for a Mine Hoist. By Graham Bright.
The Automatic Sub-station for Coal Mines. By R. J. Wensley.
Anthracite Preparation. By D. C. Ashmead.

On Wednesday morning, a technical session, in charge of Prof. James F. Kemp, was held at Irem Temple, and at the same time, a session on Mine Accounting, with R. V. Norris in the chair. The former, a joint meeting conducted by the Society of Economic Geologists, included the papers:

* Review of Lead and Zinc Mining in Pennsylvania. By Prof. Benjamin J. Miller.
* Some Unique Bolivian Tungsten Deposits. By Frank L. Hess.
General Geology of the Catorce District. By C. L. Baker.

In the session on Mine Accounting, a paper on "Underground Mine Development, Its Definition and Valuation," by J. B. Dilworth, was discussed.

After these meetings, the party was conveyed in automobiles to the plant of the Wyoming Shovel Co., at Wyoming, for luncheon. Each

* Presented by the Society of Economic Geologists. Not published by the A. I. M. E.

visitor was presented with a small army shovel, like thousands that were supplied to the American forces during the war.

Two technical sessions were scheduled for the afternoon, at Irem Temple, one on general technology, which was cancelled, and one on Economic Geology, under the chairmanship of Dr. H. M. Chance. At this session Professor Kemp lectured on the stratigraphy of the anthracite region, showing numerous lantern slides to illustrate his discussion. Other papers were:

Geology of the Namma Coal Field, Burma. By Edel Moldenke.
* Review of Deep-seated Oil and Gas in the Appalachians. By George Ashley.
* Coning of Waters in Mexican Gulf Coast Wells. By E. DeGolyer.
Petroliferous Rocks in the Serra da Baliza. By E. P. de Oliveira.

The papers scheduled for the second session of the afternoon were:

The Queen Nine-hearth Roaster. By J. Moore Samuel.
Flotation of Pyrite. By W. S. Morley.
Relation of Gypsum Supplies to Mining. By D. H. Newland.

During the afternoon many of the members visited the works of the Hazard Wire Rope Co. and the Vulcan Iron Works.

On Wednesday evening the Anthracite Section gave a dinner to the visiting members and their guests, in the large dining room of Irem Temple. R. V. Norris, chairman, concluded his brief welcome to the members and guests by saying:

"We want you to see and to know what our region is. It seems incredible that a district but a few miles from New York should be as little known and as much maligned."

He read letters and telegrams from "Eddie" Pechin, one of the founder members who is now 87 years young, William Kelly, Calvin W. Rice, and Herbert Hoover, and then announced the presentation to the Institute of an American flag made by an early member of the Institute, Mr. Lee, and carried to many meetings and foreign countries. It is the only American flag, it is believed, that has been flown at any time in the London Guildhall.

He then introduced Prof. J. F. Kemp as toastmaster, who said, in part: "Do you know, this is the best birthday party I ever attended. If I were to see a birthday cake with fifty burning candles come through that door, it would seem to me as if it were the end of the long, long, night of waiting, when all my dreams came true."

Professor Kemp introduced Doctor Drinker, "the halest and heartiest of our centenarians," who told of the first meeting at which the Institute was founded and the subsequent meetings in its early life when "the trips were looked forward to as social gatherings of friends of common interest in the promotion of the success and growth of the Institute."

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Col. Arthur S. Dwight was the next speaker. He told very interestingly of the trip of the Institute delegation to England and France on the occasion of the presentation of the John Fritz Medal to Sir Robert Hadfield and Eugène Schneider, first paying his respects to "our dear old antediluvian, as he calls himself, our dear Doctor Drinker." He continued:

"He has been all his life so modest and retiring in all the great public work that he has carried on, as educator and supporter of many of the important public movements, like the American Forestry Association, that we would never think, from the account that he has given of that great band of Founders, what he has done. However, I wish to call attention to one thing that perhaps few of you may realize, although I think you all know, and that is the great part that Doctor Drinker took in the winning of the war. I think I can truly say that Doctor Drinker himself has done more than any other person present in this room to help win the great war. He is the one who is generally recognized as the Father of the Plattsburgh System, which did so much to furnish trained officers when we were called into the great conflict.

"Also, as the chairman of the committee of this Institute to urge upon the Government the great desirability of creating an Officers Reserve Corps, which would stand ready to supplement the officer personnel of our regular Army, in the days when hostilities were declared, he did a great work in assisting in the creation of that Officers Reserve Corps, which became a fact when the National Defense Act, which was signed in June, 1916, was made a law.

"At the same time, and parallel with this activity, was the development of the Students Training Camps, summer training camps, which became so eminently efficient and popular that there came a great demand from business men and civilians in general to have the benefit of that military training, and you all know what a tremendous movement developed from that in the Plattsburgh camps and the other military camps, which gave us, when war was declared, a large and fairly well trained officer personnel. It was the one thing that permitted us so promptly to take our place in the firing line, and I think we can all feel proud that Doctor Drinker's name was coupled with that of General Leonard Wood as the fathers of that movement! Doctor Drinker's contribution to the defense of the nation was worth whole regiments of fighting men."

Colonel Dwight presented to the Institute a fraternal message from the Institution of Mining and Metallurgy in London and announced the conferring of honorary membership in that body on President Ludlow.

The toastmaster then introduced Lieut. Col. J. A. S. Ritson of Great Britain, a delegate to the Congress on Safety First.

Thursday was a perfect day overhead and even before starting time the party began to gather in front of Irem Temple ready for their 117-

mile trip through the central and southern anthracite fields, a trip that proved to be one of the most enjoyable and profitable ever offered to the members of the Institute. A guide book had been prepared on the plan of the Automobile Blue Book. All the speedometers were set at zero when starting, and as the drivers read off miles and tenths, the guests had only to turn to the guide books to learn of every interesting object that came within view. In addition, large neatly painted signs that could be read rapidly told the names of the collieries or breakers, their output and a few words of their history.

In succession, there were passed the Highland Collieries of the G. B. Markle Co. and the huge conical bank of the Drifton breaker; Jeddo No. 4 of the G. B. Markle Co., Harleigh No. 7 of the same company, and finally, Beaver Meadow, the birthplace of Jay Gould.

At 50 miles the southern region was entered, and the Nesquehoning Breaker, which produced in 1918 nearly a million tons of coal, was passed; at 60 miles, the Hauto Power Plant of the Pennsylvania Electric and Power Co. was reached, a plant that now has a capacity 60,000 kw. and is still building. Its motive power is steam generated from fine anthracite on Coxe stokers. The members of the Institute were conducted through this plant and then were taken a short distance to Greenwood Park picnic grounds, belonging to the Lehigh Coal & Navigation Co. Here luncheon was served to 500 members and guests, and a demonstration of first aid was given by three of the local teams. The return trail led through Summit Hill to Tamaqua where passengers rested at the collieries, viewing the bailing plant of the Lehigh Coal & Navigation Co., where they were using four buckets, each of a capacity of 3600 gal., each making a round trip in about 100 seconds. After crossing the Schuylkill River, the party stopped to see the stripping and mining of Mammoth and Four Foot veins of the Lehigh & Wilkes-Barre Coal Co. The total thickness of this coal is 25 feet.

PAPERS

Geology and Ore Deposits of Jerome District

By Louis E. Reber, Jr.,* Ph D., Jerome, Ariz.

(Lake Superior Meeting, August, 1920)

The town of Jerome is located in Yavapai County in north central Arizona. It has a population of over 6000 people and the two important mines of the district, the United Verde and the United Verde Extension, lie almost within its limits. Perched high on the southwest side of the Verde Valley, at elevations of 4800 and 5500 ft. (1463 and 1676 m.), both mines have long tunnels from their lower levels through which ore is hauled to their respective smelters. The two smelter towns of Clarkdale and Verde, out in the center of the valley and about 2000 ft. (609 m.) below, form part of the wide panorama that meets the eye at Jerome.

In the vicinity of Jerome, the great southern escarpment of the Arizona plateau forms the northeast side of the Verde valley. The general level of this escarpment, or "the rim" as it is called, is about 4000 ft. above the Verde River. A corresponding elevation is reached only by the highest parts of the Black Hills, which bound the valley to the southwest. The two lava-capped mesas, known as Mingus and Woodschute Mountains, form the culmination of the Black Hills. That part of the old Black Hills mining district which lies between their crests and the Verde River is now known as the Jerome or Verde copper district. However, if the smelters are not included, a strip about 7 mi. long and 3 mi. wide (11 by 5 km.) embraces practically all the properties belonging to the Jerome district. This strip lies parallel to and along the side of the Verde valley from about First View, 2 mi. northwest of Jerome, to the neighborhood of Allen Spring, near the south end of Mingus Mountain.

The rank of the district in copper production, as well as features of peculiar geologic interest in connection with the ore occurrences, entitle the district to much more consideration than it has received in the literature of copper mining. Arizona stands preëminent as a copper-producing state. In 1918, it produced four-tenths of the total amount produced in the United States, and nearly equaled the combined production of its three chief competitors, Montana, Michigan, and Utah. The production of the Jerome district places it as sixth among the copper camps of the United States; in Arizona, Jerome ranks third, being surpassed in production by both Bisbee and Globe-Miami.

* Geologist, United Verde Copper Co.

Two mines furnish practically the entire production of the Jerome district. During the period of maximum production to meet war-time demands, the two mines, the United Verde and United Verde Extension, produced approximately 7,000,000 lb. and 4,000,000 lb. of copper per month, respectively; together they contributed about one-sixth of Arizona's production. With the end of the post-war depression, this production will again be reached if not exceeded. The United Verde is also important as a producer of gold and silver. The ore is smelted direct; the production figures indicate the size and richness of the deposits.

The United Verde offers possibilities of becoming one of the deep mines of the world, and, with the deepest shaft at the 2500-ft. (762-m.) level, has already blocked out one of the largest known deposits of massive sulfides, probably exceeded in size only by those of Rio Tinto, Spain. The United Verde Extension possesses one of the largest known bodies of high-grade chalcocite ore, which is particularly noteworthy as the result of the only recorded instance of important secondary sulfide enrichment known to have taken place in pre-Cambrian time.

Jerome has been accorded surprisingly little consideration in the technical literature. In spite of the many interesting features, this is particularly true in regard to geological description, largely because of the complexity involved and the lack of systematic study. Until recently, a few private reports by consulting geologists represented most of the geological work done in the district. The United States Geological Survey has done comparatively little work there because, up to the time of the spectacular discovery of the United Verde Extension orebody, Jerome was practically a one-mine camp. (Ransome: U. S. G. S. *Bull)* 529 (1913) 192.

In a report prepared in 1916, and subsequently published in several periodicals, Provot[1] described the geology of the camp and correctly interpreted the more important structural relations. Since then, T. A. Rickard[2] has published an article on the United Verde Extension giving an outline of the geologic conditions, and J. R. Finlay has published a very comprehensive article dealing with the geological associations of the Jerome district.[3] A theory is developed as to the general relationships of the broadest kind, and some very interesting and instructive possibilities are outlined. Such broad theorizing is necessarily handicapped until the more local problems have been completely worked out.

As is often the case where pre-Cambrian formations are concerned,

[1] F. A. Provot: Geological Reconnaissance of the Jerome District, Jerome, Ariz., May 13, 1916. Abs. in *Enging. & Min. Jnl.* (Dec. 9, 1916) **125**, 1028.

[2] T. A. Rickard: The Story of the United Verde Extension Bonanza. *Min. & Sci. Pr.* (Jan. 5 and 12, 1918) **116**, 9, 47.

[3] J. R. Finlay: The Jerome District of Arizona. *Engng. & Min. Jnl.* (Sept. 28 and Oct. 5, 1918) **106**, 557, 605.

the geological record is extremely interesting, but complex and difficult to unravel. The geological literature on the district shows discrepancy as to reported facts of observation as well as in interpretation. Thus, without unlimited space for description, it is difficult always to separate the observed facts from the inferences with that clearness demanded by scientific accuracy. It is believed that the working out of the geology of the district in detail will prove of particular value in connection with general prospecting.

The problems relating to development and exploratory work in the more or less well defined mineralized zones of the producing mines are chiefly local ones, and in this case the more general problems, though important, are necessarily of secondary consideration. Geologic work of a local character related to the operation of the United Verde mine has claimed the greater part of the writer's time, and some of the most important general questions are not yet settled. The complete deciphering of the geologic record, however, will unquestionably benefit the developed mines as well as the prospects, and this end has been held in view. The correct classification and grouping of the different pre-Cambrian rock types requires all the combined resources of field study and laboratory examination, and much remains to be done. In addition to the microscopic work done by the writer, between fifty and a hundred Jerome specimens have been described by Charles P. Berkey, chiefly for the Jerome Verde Co., and A. N. Winchell has described a somewhat smaller number for the United Verde. The work of R. W. Hart, geologist for the United Verde Extension, has contributed to the general knowledge of the district.

General Geology

Probably the best way to summarize the general geology of the district is to outline the geologic history according to the most plausible interpretation. Some of the more debatable points will be discussed later.

Pre-Cambrian

The geologic record begins in the Algonkian era of pre-Cambrian time. A great greenstone complex, indicating a period of important volcanic activity, is probably the earliest record in the Jerome district. The basement formation on which these lava flows and agglomerates were deposited is unknown. Apparently younger than the greenstones is a series of clearly bedded sedimentaries which, though containing much fragmental volcanic material, are evidently water-laid and in part well sorted. This latter series records the gradual regaining of supremacy of the normal forces of erosion and slow sedimentation following the rapid and disordered outpourings of volcianc material.

Later a period of deformation, probably related to the approaching invasion of a granitic batholith, squeezed the bedded sediments into close folds trending about north-northeast and south-southwest. The greenstones in the southern end of the district have been folded along lines trending nearly east and west. Following this deformation, the region was invaded by masses of quartz porphyry, presumably an outlying phase of the Bradshaw granite, which underlies the greater part of Yavapai County to the south and west of Jerome. While still deeply buried, certain limited zones, or portions, of the quartz porphyry were rendered schistose by further deformation. After this last deformation, masses of augite-diorite, called the United Verde diorite formation, were intruded into the quartz porphyry and older rocks in the vicinity of Jerome. The ore deposits, which will be discussed later, were formed after the intrusion of the diorite.

After the formation of the large schist-replacement orebodies, but before the mineralization was entirely completed, a series of small diorite or andesite dikes cut the ore masses, the United Verde diorite, and as far as known, all other pre-Cambrian formations. These dikes are not uncommon throughout the Jerome district and probably are as widely distributed as the Bradshaw granite. They range in thickness from a few inches to about 40 ft., but most of them are from 1 to 4 ft. (0.3 to 1.2 m.) thick. In the vicinity of Jerome, they have a general east and west trend, thus cutting the local schistosity of the quartz porphyry nearly at right angles. The quartz veins of the southern part of the district were probably formed after the principal mineralization, and may be related to the same fissuring as the dikes.

The intrusion of the United Verde diorite practically marked the end of the intense deformation that affected the older rocks. Though the material of the orebodies sometimes shows evidence of granulation and brecciation and the small fine-grained dikes are in some places quite schistose, the coarsely crystalline, resistant masses of United Verde diorite show very little evidence of dynamic metamorphism. Numerous faults, for the most part of very small throw, mark the final stage of this period of deformation. The diorite-andesite dikes are displaced many times by these faults, usually only a few feet. These small faults or slips often follow the trend of the shearing in the porphyry.

The period of deformation and igneous intrusion preceding the formation of the ore deposits no doubt produced a rugged and mountainous land area, high above sea level. The erosion period required to reduce this region to the nearly perfect base level that preceded the invasion of the Cambrian sea must have been a long one. During this long period of erosion the orebodies were exposed, portions of them were worn away, and portions enriched by the development of secondary chalcocite.

Paleozoic

Even a peneplain as nearly perfect as that developed before the advance of the Cambrian sea must have sloped upward from the sea coast to the interior of the continent. The sinking of the continental mass and advance of the sea took place very slowly, and stream erosion continued in parts of the continent not yet submerged. The sea probably reached the Jerome district in middle Cambrian time. The heavy mantle of alluvium, residual soil, and altered rock that no doubt covered the land in its mature stage of erosion was swept away by the sea, and the pre-Cambrian bed rock was scoured clean. On this was deposited a thin blanket of beach sand and fine pebbles, which formed the Tapeats[4] sandstone. This varies in thickness from nothing to 100 ft. and tends to fill the minor irregularities of the pre-Cambrian surface.

More or less clayey material was deposited above the sandstone; this is now represented by 10 to 20 ft. (3 to 6 m.) of red or greenish-yellow shale. Although in some places limestone appears to overlie the sandstone conformably, it is possible that a greater thickness of shale was formed and later eroded or that some of the limestone is of Cambrian age. The sandstone is of a maroon color, and often highly ferruginous. The iron is perhaps derived from the gossan of the pre-Cambrian orebodies.

Overlying the basal sandstone are from 300 to 500 ft. (91 to 152 m.) of limestone, all or in part of Devonian age, from 300 to 500 ft. of limestone of Mississippian or lower Carboniferous age, and from nothing to 500 ft. of red sandstone and shale of Permian age.[5] The formation of the deposits of each of these four periods, Cambrian, Devonian, Mississippian, and Permian, was preceded and followed by periods of uplift and erosion. Much greater thicknesses were no doubt originally present, probably including some Pennsylvanian or upper Carboniferous limestone.

The lower limestone, in so far as of Devonian age, corresponds to the Temple Butte limestone of the Grand Cañon section. Ransome[6] has already applied the names Redwall limestone and Supai formation to the Mississippian limestone and overlying red sandstone and shale of the Verde valley.

When a great area of limestone is uplifted slightly above sea level and later submerged and limestone deposition is continued, an important hiatus

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[4] F. L. Ransome: Some Paleozoic Sections in Arizona and their Correlation. U. S. Geol. Sur. *Prof. Paper* 98–K (1916) 159–162.

[5] This red sandstone and shale, known as the Supai formation, together with the overlying Coconino sandstone and Kaibab limestone of the Grand Cañon section were, until recently, classed by the United States Geological Survey as of Pennsylvanian age.

[6] *Op. cit.*

in the sedimentary record may be inconspicuous and difficult to find. Thus at Jerome the breaks delimiting the Devonian limestone have not been recognized. However, in a section examined less than a mile northeast of Jerome, the lower 300 ft. of the limestone is nearly all somewhat dolomitic; most of it weathers to a brownish color, and it contains many sandy beds. Above this the limestone is white, non-magnesian, and free from sand. This change probably marks the top of the Devonian, though Ransome[7] provisionally assigns 500 ft. of limestone to the Devonian at Jerome. The lower part of this white limestone is fine-grained and banded with chert; fossils are rare. The transition to the heavy-bedded highly fossiliferous material above is, however, gradual.

The uplift that brought the deposition of the Carboniferous limestone to a close was evidently associated with important deformation in adjoining regions. Rapid deposition of clastic material persisted for some time after the advance of the "Permian sea," followed later by limestone deposition. Along the northeast side of the Verde valley opposite Jerome, the Permian rocks correspond very closely to those found in the Grand Cañon 80 mi. (128 km.) north. These include over 1000 ft. (305 m.) of the red sandstone and several hundred feet of heavy-bedded white sandstone, overlain by a thick limestone formation. Thus it is probable that the complete series was deposited over the Jerome district, though only 500 ft. of the red sandstone shows in the side of the valley west of Jerome, and this thins to nothing farther south.

Mesozoic

There is no further record of marine sedimentation near the Jerome district, although it is possible that the Mesozoic formations, which are extensively developed in the northeastern part of the state, originally extended as far west as Jerome. They no doubt covered much of the eastern part of the state.

The post-Cambrian formations lie practically flat in the Jerome district, except close to the faults. Farther north, dips of a few degrees to the north or east prevail, indicating slight folding, probably in late Mesozoic or early Tertiary time.

Cenozoic

Tertiary.—It is certain that there has been no marine invasion since the close of Mesozoic time. The record is one of volcanic activity, terrestrial deposition, and faulting. The volcanic field of the San Francisco Mountains is not far north of the Jerome district. The work of

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[7] *Op. cit.*

Robinson[8] helps to interpret the history of the adjoining regions. Robinson has called attention to a peneplain, or relatively even surface, developed by stream erosion during the Tertiary period, upon which the late Tertiary lavas were poured. In the Jerome district, the irregularities of the surface beneath the volcanics suggest that there was some uplift and dissection of the peneplain before the lavas were deposited. The Paleozoic formations were trenched with steep-sided gulches, and the Verde valley seems to have existed somewhat along its present lines. Many of the gulches were more or less filled with poorly stratified gravels and boulder material before being covered with lava. This gravel deposition was due to some change in the condition of drainage, perhaps the damming of the streams by the first lava flows.

The great outpouring of basaltic lavas that at one time covered a large part of northern Arizona is assigned by Robinson to the latter part of the Pliocene epoch; these lavas were very fluid and spread out for long distances. It seems probable, however, that there were many widely scattered vents, a large number of which never developed conspicuous cones. Many of the cracks now occupied by dikes were feeders of this great lava field. No doubt most of the outlying patches of lava were once connected with the main field. The hills above Jerome are still capped with a layer of this lava about 700 ft. (213 m.) thick. The younger lavas of the San Franciscan volcanic field do not seem to be present at Jerome.

Quaternary.—Where the lavas flowed across the Tertiary Verde valley below Camp Verde, about 30 mi. (48 km.) southeast of Jerome, a dam was formed, which resulted in a lake at least 35 mi. long and 6 or 8 mi. wide. In connection with reconnaissance mapping for a geological map of Arizona, O. P. Jenkins, the geologist of the State Bureau of Mines, has recently studied and mapped the limestone deposits formed by this lake and has named them the Verde formation.

Following the outpourings of basaltic lava, came a period of normal faulting, which uplifted the western side of the Verde Valley more than 2000 ft. (609 m.), possibly as much as 5000 ft. The trend of the principal breaks developed at this time varies from north and south to northwest and southeast. It is probable that all of this faulting took place after the Tertiary lavas were formed, and was more or less contemporaneous with the deposition of the lake beds of the Verde formation. The lake beds are displaced by the faulting in one or more places, and the great thickness of these beds suggests faulting during deposition. At the United Verde Extension smelter, drill holes have penetrated over 1200 ft. of material, all of which probably belongs to the Verde formation. A simi-

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[8] H. H. Robinson: The San Franciscan Volcanic Field, Ariz. U. S. Geol. Sur. *Prof. Paper* 76 (1913).

lar thickness of the Verde formation shows in the valley above the collar level of the drill holes.

The interrupted drainage of the lake period and the relief produced by the faulting promoted the formation of extensive deposits of poorly sorted gravel and boulders differing from those underlying the lava chiefly in the presence of abundant fragments and boulders of basalt. These younger gravels merge into the margins of the lake beds in some localities. Following the restoration of normal drainage, the present channel of the Verde River and numerous tributary gulches have trenched the Verde formation and the gravel deposits.

The uplifted scarp of one of the principal faults was especially subject to erosion, so that a strip of the pre-Cambrian has been exposed along the southwest side of the Verde valley, trending in a southerly direction from a point just north of the town of Jerome. This fault, known as the Main or Verde fault, passes through the town of Jerome and practically divides the district into two parts. East of this fault, the pre-Cambrian formations are deeply buried.

The Pre-Cambrian Formations

Greenstone Complex

Although including a number of different rock types, the greenstone complex is a unit in regard to thorough dynamic metamorphism and prevalent green color. Much of the rock, particularly of the fragmental phases, is as siliceous in composition as the light-colored quartz porphyry, yet carries enough chloritic material to give it a green color when fresh and an iron-stained appearance when weathered. Part of the metamorphism may be the result of a period of deformation earlier than that connected with the intrusion of the Bradshaw granite.

The non-committal term, greenstone, is a useful one in connection with much of this material, the exact nature of which it is difficult to determine. Many of these rocks are identical with those to be found in the pre-Cambrian of the Lake Superior region. It is believed that these greenstones are largely surface volcanics, although many small intrusions are recognizable; and it is possible that some larger masses may be found to be of intrusive origin. In some localities, lava flows are definitely recognized; and elsewhere, large areas of fragmental volcanic material are unquestionably present. In Mescal gulch, near the Pittsburgh-Jerome property, there are some interesting exposures similar to those of the Deer Lake agglomerate found north of Marquette, Mich. The fragmental material sometimes includes distinctly water-worn grains or pebbles, although more than a suggestion of sedimentary stratification is rare. Included in the greenstone complex are small quantities

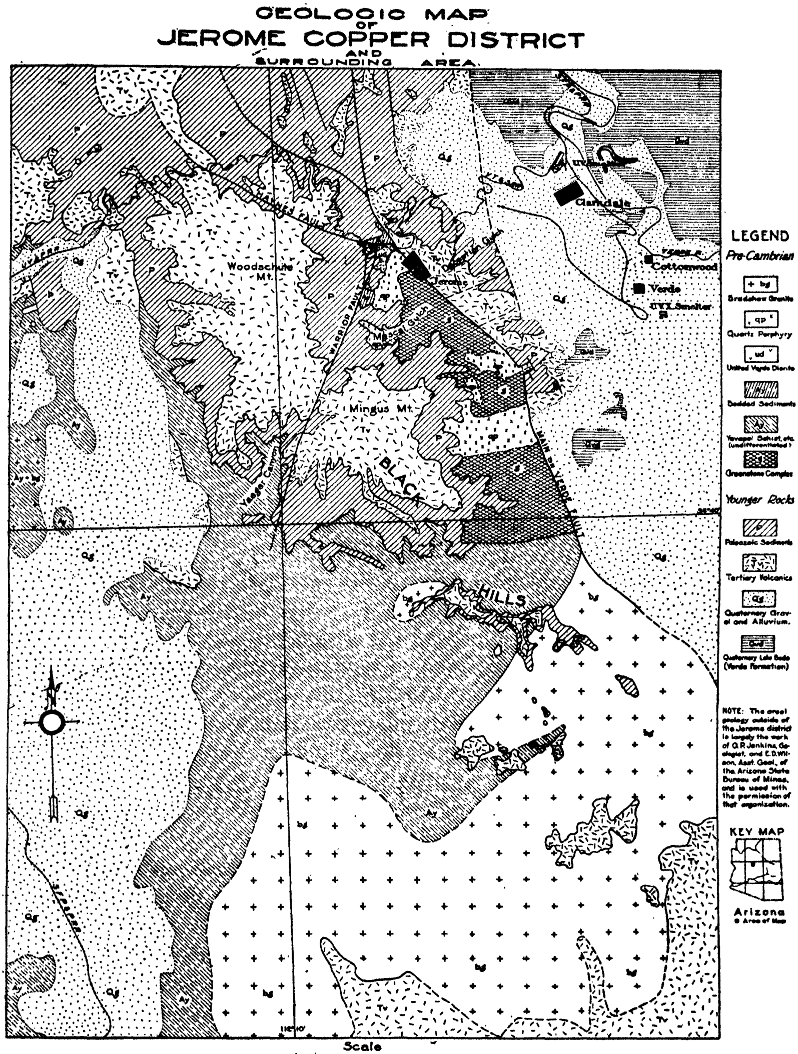
GEOLOGIC MAP
OF
JEROME COPPER DISTRICT
AND
SURROUNDING AREA
Woodachute Mt.
Mingus Mt.
BLACK
HILLS
Clarkdale
Cottonwood
Verde
Jerome
WARRIOR FAULT
MAIN VERDE FAULT
LEGEND
Pre-Cambrian
Bradshaw Granite
Quartz Porphyry
United Verde Diorite
Bedded Sediments
Yavapai Schist, etc. (undifferentiated)
Greenstone Complex
Younger Rocks
Paleozoic Sediments
Tertiary Volcanics
Quaternary Gravel and Alluvium.
Quaternary Lake Beds (Verde Formation)
NOTE: The areal geology outside of the Jerome district is largely the work of G. R. Jenkins, Geologist, and E. D. Wilson, Asst. Geol., of the Arizona State Bureau of Mines, and is used with the permission of that organization.
KEY MAP
Arizona
Scale
Miles

FIG. 1.

of light-colored fragmental material, probably in large measure of volcanic origin.

In the area along the Main fault where the pre-Cambrian is exposed, the greenstone complex is the predominant formation from near the Copper Chief mine, in the south end of the district, north to the outskirts of the town of Jerome; see Fig. 1. In Deception gulch, at the north end of this area, where it adjoins the quartz porphyry, the nature of the material is open to question. The appearance is similar to that of known volcanic fragmentals and locally there are some obscure indications of its fragmental character. The microscopic evidence is conflicting but seems to point in the same direction. Nevertheless, the rock has much the appearance of an intrusive igneous rock and Finlay has named it the Deception porphyry. The nature of its contact with the quartz porphyry indicates that one or both of the rocks must be intrusive.

Bedded Sedimentaries

Geologists of the United States Geological Survey[9] have mapped and described the Bradshaw quadrangle, the north end of which is 18 mi. south of Jerome. Yavapai schist and Bradshaw granite are shown to extend across this line into the Jerome quadrangle. The Bradshaw granite exposure is continuous to within 6 mi. of Jerome, while the exposure of pre-Cambrian rocks is continuous to Jerome.

As might be expected from their relation to the Bradshaw granite, the dynamic metamorphism of the schists has been, in general, more intense throughout the Bradshaw quadrangle than farther north. Although the Yavapai schists have been determined to be largely of sedimentary origin, nowhere in the Bradshaw quadrangle is sedimentary bedding more unmistakable than in the bedded sedimentaries at Jerome. These rocks are characteristically more massive than schistose, though closely folded; however, they probably correspond closely enough in composition and general appearance with parts of the Yavapai schist, to the south, to be included in that formation, though preferably as a distinct member. On the other hand, Yavapai schist, as defined by the Survey geologists, does not seem to include large masses of greenstone of predominantly volcanic origin, such as the "greenstone complex" of the Jerome district.

The bedded sediments of the Jerome pre-Cambrian are closely folded and the dip of the beds is steep. Minor folds, as a rule, are not conspicuous except in some of the more siliceous bands, which sometimes show remarkable crenulations and drag folds. A great variety of material is included, from very fine-grained quartzite and slates of various kinds to thick-bedded conglomerates. These conglomerates

[9] T. A. Jaggar, Jr. and Charles Palache: U. S. Geol. Sur. *Folio* 126, *Bradshaw Mountains Folio* (1905).

typically consist of more or less abundant well rounded pebbles in a tuffaceous matrix including numerous well preserved feldspars, giving the appearance of a feldspar porphyry where the pebbles do not show. Also a bed, or lens, of fairly pure crystalline limestone or marble 20 ft. thick has been observed.

Near Walnut Spring, west of Jerome, a volcanic agglomerate consisting largely of quartz porphyry occurs between the more definitely bedded material and the main porphyry mass. This is clearly a fragmental, not an intrusive breccia, and it is extremely difficult to draw the line between the fragmental and the intrusive quartz porphyry.

Quartz Porphyry

At the south end of the Jerome quadrangle, the Bradshaw granite is coarse-grained, but becomes progressively finer grained to the north. The marginal phases often differ from the usual type. The quartz porphyry of the south end of the Jerome district is, in general, of a more granitic character than that near Jerome. It is believed that further work will prove a progressive change from the fine-grained granite of the known Bradshaw exposure to the typical rhyolitic porphyry of the north end of the district.

Though many phases of the quartz porphyry are represented, it is believed that they are all of essentially the same age and may be considered as a unit, except for the so-called Deception porphyry, which may prove to be older. The usual composition is that of a normal rhyolite, though often an albite rhyolite and rarely a dacite. The color is usually light, though dark gray or green phases are not rare. The quartz and feldspar phenocrysts vary both in actual and relative abundance and to some degree in size, while the fineness of the ground mass is also variable. Perhaps the most difficult feature to account for is the variation in degree of dynamic metamorphism; however, the main mass of the Bradshaw granite also shows this variation to some extent. Some portions of the quartz porphyry show extreme dynamic metamorphism and some show practically none.

Mine workings under the cover of Paleozoic sediments east of the Main fault have shown that the quartz porphyry predominates throughout a large area in the northeastern part of the district. This quartz porphyry is for the most part very massive, particularly on the east side. Just east of the fault and extending south from the United Verde Extension orebody and showing in Jerome Verde workings is a relatively limited zone where the quartz porphyry is quite schistose. In the southern part of the district, the quartz porphyry is generally massive with a few schistose areas of limited size. The largest and most important area of schistose quartz porphyry is that of Cleopatra hill and the United

Verde mine. To the east, between this area and the fault, the rock is relatively massive. Though the quartz porphyry of the United Verde and the United Verde Extension and Jerome Verde must have been continuous before the faulting, the schistose areas appear to have been distinct.

The evidence now available is not sufficient to make it absolutely certain that there is not more than one distinct quartz-porphyry intrusion or that the most schistose of the porphyry does not belong with the quartz-porphyry fragmentals. In either case, changes in the historical outline would be required and if the assumed relation to the Bradshaw granite proves incorrect, the pre-Cambrian chapter must be revised.

United Verde Diorite

Diorite of two distinct ages is described as occurring in the Bradshaw quadrangle.[10] The first varies from a quartz diorite to a gabbro, and is believed to be intimately related to the Bradshaw granite; the other is a gray quartz diorite distinctly younger than the granite. Even the younger quartz diorite, however, may reasonably be looked upon as a very late phase of the granite intrusion. In the Grand Canyon, similar diorite is found which is intruded by granitic pegmatites.[11]

The United Verde diorite is an augite diorite with diabasic phases and evidently belongs with the older diorite, though showing less dynamic metamorphism than is usual farther south. Thus, it is believed that the United Verde diorite is also related to the Bradshaw granite. The principal mass of this diorite is about ¾ mi. long and ¼ mi. wide (1.2 by 0.4 km.). It trends northeast and southwest and forms the hanging wall of the United Verde ore zone. Farther north, another mass of this diorite is partly exposed. In the United Verde Extension workings, just east of the fault, more of this diorite is exposed, which probably was once a part of the large mass on the west side of the fault. To the southeast, the end of another mass is blocked out by Jerome Verde workings. Several miles to the southwest, similar diorite shows in Yaeger Canyon; it has not been identified in the southern part of the district. Practically no evidence of contact metamorphism has been observed along either the quartz porphyry or the United Verde diorite contacts.

Ore Deposits

Three distinct ore deposits have contributed the copper production of the Jerome district. In order of importance, they are that of the United Verde, the United Verde Extension, and the Copper Chief-Iron

[10] T. A. Jaggar, Jr. and Charles Palache: *Op. cit.*

[11] L. F. Noble and J. Fred Hunter: A Reconnaissance of the Archean Complex of the Granite Gorge, Grand Canyon, Ariz. U. S. Geol. Sur. *Prof. Paper* 98-I (1916) 102.

King. The Maintop orebody, from which the Jerome Verde Co. has been shipping ore during the last two years, though more or less distinct, may be looked upon as a part of the United Verde Extension ore zone. The Copper Chief-Iron King deposit has been more important as a producer of gold and silver from oxidized gossan material, although a considerable tonnage of sulfide copper ore from the Iron King claim was smelted years ago by the Equator Mining Co. The ore zones of each of these three deposits are distinct from the others, but it is believed that they are closely related as to character and origin.

About thirty prospecting companies have been active in the Jerome district during the past few years, and some have spent large sums of money. In view of the expenditure, the results have been disappointing, though some ore showings have been found. These companies have been exploring ground where promising surface showings were clearly lacking or where the pre-Cambrian surface was covered by younger formations.

The United Verde and the Copper Chief-Iron King ore deposits lie west of the Main fault in the area of exposed pre-Cambrian rocks, the former at Jerome, and the latter about 4 mi. to the south. Since the uncovering of these orebodies by the post-fault erosion, conditions apparently have not been favorable to secondary sulfide enrichment. Probably the steep slopes and rapid erosion favored the scattering rather than the concentration of the copper. Oxidation of the sulfides with little copper enrichment has preceded the actual erosion surface, and the enrichment of the pre-Cambrian erosion period has been in large measure destroyed instead of increased. Some chalcocite remains in the upper levels of the United Verde mine, but the localities are now inaccessible. H. V. Winchell suggests that a higher zinc content may have rendered these deposits less susceptible to secondary enrichment than that of the United Verde Extension. In the United Verde Extension, to the east of the fault, the oxide and chalcocite zones of the pre-Cambrian period of erosion and enrichment are still protected by the overlying blanket of Paleozoic sediments, and the zone of altogether primary sulfides has not been reached in the orebody. Thus, in the United Verde Extension mine, only the oxide and chalcocite zones can be studied and in the United Verde and Copper Chief only the oxide and primary sulfide zones. Nevertheless, it is clear that they have much in common.

In each case large masses of sulfides have been developed by the replacement of schists. In all three pyrite predominates. Chalcopyrite is known to be in two and in the United Verde Extension it appears to have been the chief primary copper mineral. In all three the primary sulfides carry gold and silver and are associated with masses of very distinctive jaspery quartz. The fact that the ore deposits of the two larger mines are not far apart and seem to be of almost the same age, also indicates that they have a common origin.

The Copper Chief mass differs from the others in that, so far as known, it has only a small continuation with depth. However, the sulfides are distinctive and identical in structure and composition with parts of the sulfide mass in the United Verde.

Replacement

These ore deposits, therefore, are of the schist-replacement type. The evidence of replacement is abundant and convincing. Where massive sulfides adjoin schists there is usually a gradual transition from unreplaced to completely replaced schist. More rarely, the contact is clean-cut, yet it shows all the embayments and irregularities characteristic of the walls of limestone replacement deposits. The schist structure is often strikingly reproduced in a banding of the sulfide minerals or by residual shreds of unreplaced material. Throughout the primary sulfide masses at least a suggestion of the structure of the replaced schists is universally present. Similar evidence is present in the United Verde Extension ore deposit, though somewhat masked by the additional changes due to the chalcocite enrichment.

The material that has been replaced included a wide range. The schistose condition seems to have been the most important prerequisite. In the United Verde Extension and in the Copper Chief-Iron King Ore zones, schistose quartz porphyry has been the material most abundantly replaced, with lesser amounts of greenstone schist. In the United Verde ore zone, some schistose quartz porphyry and some greenstone schist have been replaced, but a black chloritic schist not known to occur in the other two mines has been very much the most favorable to replacement. This schist,when unaltered, is dark green to black in color, of fine even texture and often resembles a black slate. The microscope shows it to be composed of an even felt of nearly isotropic chloritic material. It has the chemical composition of a ferruginous chlorite. Sometimes the black chlorite appears to replace other material and it is believed that the black schist is a product of alteration by magmatic solutions, which may have been related to the same period of igneous activity as the ore deposits. It seems clear that the chlorite was present in advance of the sulfides. In the United Verde, the black schist appears to belong with the pre-Cambrian bedded sediments. It also occurs in parts of the greenstone complex and wherever found is favorable to mineralization. The relatively massive United Verde diorite has not been affected to any appreciable extent by the mineralizing solutions.

Composition

Oxide Zone.—The material of the oxide zone is predominantly red and yellow iron oxide with more or less granular quartz and some clay. The

purer iron oxide sometimes forms lumps and in some cases the quartz acts as a cement, but it is unusual for the gossan to be massive except where the primary jaspery quartz predominates. Often this is merely brecciated and recemented by secondary quartz. In general, masses formed chiefly of secondary quartz are rare but occur to some extent in the United Verde Extension. Possibly this difference is significant in connection with the fact that the gossan of the United Verde Extension was formed under conditions favoring secondary enrichment while the others were not.

The small gold and silver values of the primary sulfides are concentrated sufficiently in the oxide material so that much of it is of commercial grade. The processes of oxidation increase the gold-silver ratio about five times, from about 1/75 to about 1/15 by weight. Sometimes a thin layer of high-grade silver ore in which native silver is conspicuous immediately overlies the sulfides.

The oxide ore of the United Verde carries about 1 per cent. copper, mostly in the form of finely divided cuprite. Copper carbonates are unusual in the gossan material though some of the copper lost from the United Verde sulfide mass is probably to be found in the small carbonate deposits in the limestones and Tertiary conglomerate to the east of the fault. Some of the limestone ore was mined from the Jerome Verde shaft in the early days and the Dundee-Arizona Co. has been shipping some of the conglomerate. It is said that copper carbonates were important locally in parts of the United Verde mine now inaccessible.

Small portions of the Copper Chief-Iron King oxide mass contain copper carbonates and some portions are high in lead. Both copper and lead carbonate ore have been shipped from the Copper Chief. The presence of lead in the gossan at the Copper Chief was difficult to understand until it was discovered that sulfides similar to those of the Copper Chief in the United Verde often carry traces of galena. Copper carbonates are practically absent from the gossan material of the United Verde Extension mine. Presumably this gossan material was formed before the Paleozoic limestones were laid down. The relative abundance of carbonates in the oxide material developed in post-Paleozoic time at the United Verde and Copper Chief is evidently due to the influence of the limestones on the character of the ground-water solutions.

The oxides of the United Verde Extension also differ from those formed more recently in that a much smaller proportion carry commercial quantities of gold and silver. Perhaps the Paleozoic limestones are also responsible for this difference, although the known erratic distribution of the precious metal values in the primary sulfides may be a sufficient explanation. If the presence or absence of overlying limestones is a significant factor, it still does not prove that the overlying formations contributed gold and silver to the oxide material. The difference may be

due to change in the chemical character of the ground-water solutions. A concentration of the values in the primary sulfides is believed sufficient to account for the oxide values.

Chalcocite Zone.—The sulfide ore of the United Verde Extension consists chiefly of chalcocite and pyrite and varies from practically pure chalcocite to material that is largely pyrite. More or less quartz is often present. Small quantities of chalcopyrite, which seem to be secondary, and a little bornite have been observed. The precious-metal values are somewhat higher in the chalcocite ore than in the primary ore.

The great mass of gossan material from which the copper has been almost entirely removed, which overlies the chalcocite, the typical "sooty" character of part of the sulfide, and the associated kaolinization, together with the microscopic evidence, are sufficient to establish the fact that the chalcocite is of secondary origin.

As has been pointed out,[12] the microscopic relation between the pyrite and the chalcocite corresponds to that most commonly exhibited between the pyrite and chalcopyrite of the primary ore in the United Verde. Thus, it would seem that the chalcocite has been developed largely from chalcopyrite rather than from pyrite and the primary sulfides should be about one-half as rich in copper as those in the secondary zone. As would be expected, however, there is evidence to indicate the replacement of some of the pyrite.

Throughout a small area, more or less distinct from the principal mineralized area of the United Verde, where disseminated pyrite seems to have been more abundant than usual in the porphyry and the rock unusually schistose, a mass of low-grade ore has been blocked out. This is valuable because it provides the additional siliceous material required in the smelting operations. The copper values appear to be almost entirely due to the secondary enrichment, and extend from near the surface to below the 600-ft. level. At present, oxidizing conditions prevail to the bottom of the chalcocite. This suggests that the chalcocite was formed before the relative uplift of the area by faulting. As the only previous erosion of the porphyry was in pre-Cambrian time, the secondary sulfide enrichment was probably pre-Cambrian.

The vertical distance from the bottom of the Cambrian sandstone to the lowest important chalcocite in the United Verde mine is about 800 ft. There is reason to suppose that the chalcocite zone there was at one time comparable to that of the United Verde Extension. If so, it must have been developed in pre-Cambrian time. The evidence of the pre-Cambrian age of the chalcocite in the United Verde Extension mine is satisfactorily conclusive.

The pre-Cambrian dikes that cut the United Verde ore deposit, the

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[12] Marion Rice: Petrographic Notes on the Ore Deposits of Jerome, Ariz. *Trans.* (1920) **61**, 60–65.

similarity of the three occurrences described, the lack of similar primary mineralization in the Paleozoic formations, the ferruginous material in the Cambrian sandstone, and the apparent truncation of the United Verde Extension deposit by the base of the Cambrian establish the pre-Cambrian age of all three schist replacement deposits.

Granting the pre-Cambrian age of the mineralization, the United Verde Extension ore deposit must have been exposed to the erosion period that preceded the deposition of the Cambrian sandstone; since that time, it has remained deeply buried. It is highly probable that a large part of that erosion period was favorable to secondary enrichment. There is no reason to suppose that the processes of late pre-Cambrian time differed materially from those of Paleozoic time. The composition and structural relations of the oxide and sulfide zones are precisely what would be expected if they were developed by erosion on the pre-Cambrian surface. Overlying the main chalcocite body, the oxide material extends about 450 ft. in a vertical direction below the Cambrian sandstone.

If the secondary sulfide enrichment that developed this chalcocite did not take place in pre-Cambrian time, it must have been accomplished while the deposit was deeply buried. It is conceivable that small quantities of oxide material and chalcocite might have been developed since the burial of the deposit. That the enormous masses of typical oxidized gossan material and chalcocite which are present could have been so formed appears altogether impossible. We have then an important example of secondary enrichment that took place in pre-Cambrian time.

At the Copper Chief, there is a layer, a few inches in thickness, of black sooty chalcocite and disintegrated pyrite overlying the primary sulfide in some places; this material carries high silver values. This slight enrichment may be related to the present oxidation.

Extensive kaolinization has affected the rocks adjoining the masses of chalcocite in the United Verde Extension. The original character of the rocks is often completely masked. Alunite, a mineral often found associated with secondary sulfide enrichment, is not conspicuously developed in this instance.

The small andesite dikes that cut the primary sulfide masses in the United Verde are affected by kaolinization similar to that related to the chalcocitization. This alteration extends to the deepest levels of the mine, and has been referred to by previous observers as necessarily related to the primary mineralization. This alteration seems to be growing weaker in the lower levels, and conditions have been peculiarly favorable to a deep-seated ground-water circulation since the faulting caused the relative uplifting of the southwest side of the valley. Gypsum crystals, evidently deposited from ground-water solutions, are not unusual in the lowest levels of the mine. Very similar alteration

has resulted from the mingling of surface water with sulfur gas from the burning fire stopes. The burning of sulfides may be looked upon as surface oxidation speeded up. Thus, it is believed that the kaolinization of the dikes is the result of an unusually deep circulation of the acid solutions resulting from surface oxidation. However, so far as known, no chalcocite has been developed in connection with this alteration.

Primary Sulfide Zone.—Quartz, pyrite, chalcopyrite, sphalerite, specularite, dolomite, calcite, bornite, tetrahedrite, and galena have been recognized as belonging to the primary mineralization. No doubt further microscopic and chemical study will add to this list. The main sulfide masses are composed predominantly of pyrite, while quartz is, in general, most abundant in the peripheral areas. In addition to the more siliceous character of the marginal phases, in the United Verde all the larger masses of relatively pure quartz occur between the massive sulfides and the diorite which forms the hanging wall of the mineralized area.

Some of this quartz nearly always separates the sulfides from the diorite. This is the characteristic jaspery quartz, masses of which occur in many parts of the district. It consists of medium or fine-grained jaspery quartz, usually gray, red, or black in color, and usually contains numerous very small specularite veins, and sometimes shows a scattering of small pyrite cubes. In the United Verde, it is intimately related to the sulfide mass with which it is associated in three ways. Usually this quartz is brecciated and cemented by sulfides, occasionally brecciated fragments of the massive sulfides are included in what appears to be the same quartz, and occasionally there seems to be an actual graduation from one to the other. These relations, together with the position of the quartz against the diorite hanging wall, where replacement would be expected to start, indicate that the development of the quartz was an early phase of the ore mineralization.

Similar quartz often occurs in lenticular masses in the green schists associated with very little sulfide matter; it was at first believed that they were formed with the greenstones after the fashion of the Lake Superior jaspilites. Banded, hematitic, iron-stained, and often highly folded jasper has been found with the bedded sedimentaries near the Jerome district, which is identical in appearance with the banded jaspilite of the Lake Superior region. However, the jaspery quartz of the lenses in the greenstone differs materially from the known sedimentary jasper and is identical with that associated with the sulfide mineralization in the United Verde. This jaspery quartz is associated with all three of the schist replacement ore deposits, particularly in the United Verde where there is probably more of this quartz than in all the rest of the district. In addition, the great majority of the masses not associated with sulfide mineralization appear to be related either to United Verde

diorite or quartz porphyry contacts. Thus, it is believed that this jaspery quartz is, in general, a phase of the ore mineralization, but a more widespread phase than that which developed the bulk of the sulfides.

Nevertheless, masses of jaspery quartz of seemingly identical appearance with those believed to be related to the ore mineralization in the Jerome district are also found in the Lake Superior region associated with greenstones, which are very similar to those of the Jerome district. This similarity of appearance and association is sufficiently striking to make one hesitate in expressing too positive a conclusion regarding the origin of much of this quartz. Though related to the ore deposition, the jaspery masses at the surface, where clearly unaccompanied by gossan indicating important sulfide mineralization, would be very unlikely to show more sulfide matter at depth. This fact has been demonstrated by a good deal of fruitless exploratory work.

The sulfide phase of the primary mineralization developed large bodies of nearly massive sulfides in which pyrite greatly predominated. Much of this material has the appearance of pure sulfides, but nearly all of it contains at least microscopically intergrown quartz. There are all gradations to the phases in which quartz predominates, which are most often found near the margins of the sulfide masses.

Chalcopyrite is the most important and, next to pyrite, the most abundant of the primary sulfides. It occurs with the other sulfides in all proportions. Where there is sufficient chalcopyrite, the sulfides are mined as copper ore. The shape of the richer portions of the sulfide mass seems to have been influenced by the structure of the schists that the sulfides have replaced. All the massive sulfides may be considered low-grade ore, although only a small portion, less than one-fifth, is of commercial grade under present conditions. No doubt part of the primary sulfide mass of the United Verde Extension is not of commercial grade though it is probable that a larger proportion of commercial ore will be found than in the United Verde mine. The United Verde sulfide mass is, however, much larger.

Sphalerite is the third sulfide in abundance though much less abundant than the chalcopyrite. The United Verde ore averages less than 2 per cent. zinc. Throughout most of the less siliceous portion of the sulfide mass, sphalerite occurs only as occasional thin lenticular bands corresponding to the former schist cleavage. Traces of galena are often found with the sphalerite in these bands. Most of the sphalerite occurs where the material is rather siliceous, in banded material with pyrite and chalcopyrite or almost alone with fine-grained quartz. In this siliceous material, the sphalerite is not accompanied by galena.

The primary sulfides average from 1 to 2 oz. of silver and from 0.02 to 0.04 oz. of gold per ton. The highest silver values are found with the sphalerite and quartz.

The specularite occurs locally intergrown with pyrite and quartz and in much of the quartz where pyrite is rare. The pyrite, chalcopyrite, and sphalerite are intergrown in such fashion as to indicate their essentially contemporaneous development. These earlier sulfides and quartz, however, are in many places full of tiny cracks filled with chalcopyrite of a later phase of the mineralization. Evidence of any replacement of the earlier sulfides by the later has not been found. This later chalcopyrite is often associated with intergrown dolomite or calcite and quartz, sometimes with a little sphalerite, and very rarely with tetrahedrite. Sometimes the intergrown quartz and carbonate form considerable masses without much sulfide. Rarely primary bornite is found, which probably belongs with this phase of the mineralization.

It is believed that the brecciation and later chalcopyrite mineralization has not been very significant in determining orebodies in the massive sulfides, but it is probable that most of the orebodies in the adjoining masses of the black chloritic schist were the work of this later mineralization. The typical schist orebodies do not usually include much pyrite, and the chalcopyrite is associated with the intergrown quartz and carbonate. The andesite dikes that cut the orebodies have been affected by this late phase of the mineralization in rare instances.

It has been suggested that solutions related to the intrusion of these dikes may have brought about the redeposition and concentration of the chalcopyrite. The writer has found very little evidence in support of this view. Possibly there is some relation between the occurrence of the dikes and of the latest and richest phase of the schist-replacement mineralization. This has not been demonstrated, however, and the andesite dikes are found in many localities where there is very little copper mineralization.

The fissure veins, such as those developed on the Brookshire and Grand Island properties, are related to this phase of the mineralization or to a still younger one. The vein filling consists of quartz, siderite or ankerite, pyrite, chalcopyrite, specularite, and tetrahedrite, with good gold and silver values. The structure is coarsely crystalline and there is no banding.

On the Shea property, a vein is being explored that is composed of quartz, pyrite, arsenopyrite, siderite or ankerite, and tetrahedrite, with traces of bornite. The tetrahedrite carries high silver values, and some ore has been blocked out. The relation of this vein to the others is not known.

There appear to have been three closely related, yet more or less distinct, phases of the mineralization that developed the schist-replacement orebodies. The first and most widespread formed the jaspery quartz. The second formed the massive sulfides, including most of the chalcopyrite. The third added somewhat to the richness of the

massive sulfides and was largely responsible for the outlying orebodies in the schist. The difference between the central and outlying portions of the massive sulfides of the second stage of mineralization may have been due to higher temperatures in the central portions. The sericitic alteration of the quartz porphyry and the development of the black chlorite are the only rock alterations that may be related to the primary mineralization. The relationship, however, does not appear to be a close one.

Structural Relations

United Verde.—The structural conditions that determined the localization of the United Verde mineralization seem to have been particularly favorable. The concave margin of the United Verde diorite formed a steeply pitching, inverted trough of relatively impervious material, such as would tend to draw together and localize the deep-seated solutions in their upward course.

The United Verde diorite was intruded approximately along the contact of the quartz porphyry and the pre-Cambrian bedded sediments with a general northeast-southwest trend. It is the quartz-porphyry side of the old contact between the porphyry and the bedded schists that lies on the under, or southeast, side of the diorite. However, because of irregularities in the old contact and the curve in the diorite contact, this curve was largely occupied by a mass of the pre-Cambrian sediments. Also, this curve in the margin of the diorite wraps around the end of the most schistose portion of a large schistose area of the quartz porphyry. The sedimentary material under the diorite trough was largely altered to the black chloritic schist. Thus, the diorite wall favored the localization of the solutions; the diorite contact and the permeable quartz porphyry-sericite schist favored the penetration of the solutions; while the less permeable but most replaceable black schist favored substitution.

The greater part of the material under the curve in the diorite hanging wall, including some of the schistose quartz porphyry as well as the black chloritic schist, was completely replaced by a large irregular mass of quartz and sulfides having a form like the filling of an extremely irregular pipe, a pipe with each cross-section somewhat different from the next. Tongues of the black schist, projecting to the south into the porphyry, which were not completely replaced were more or less mineralized.

United Verde Extension.—The main sulfide mass of the United Verde Extension ore deposit seems to have an irregular pipe-like form similar to that of the United Verde. This mass is almost entirely surrounded by quartz porphyry and seems to have been localized by a fissure or shear zone nearly at right angles to the normal trend of the schistosity of the very schistose porphyry.

A sulfide mass that is probably a tongue projecting to the north from the main sulfide body lies against a hanging wall of United Verde diorite; this diorite may have had some structural influence on the main body. Most of the sulfide body of the United Verde Extension carries pyrite and was, before enrichment, a pyrite-chalcopyrite intergrowth corresponding to the principal sulfide mineralization phase in the United Verde; therefore, it could not have been entirely the work of the youngest or chalcopyrite phase. Thus, if the east-west fracturing or shearing in the United Verde Extension preceded the mineralization, it cannot be correlated with the formation of the breaks occupied by the andesite dikes. There is no indication of pre-mineral cross fracturing in the United Verde.

There appears to have been a local change in the direction of the schistosity of the quartz porphyry where the main sulfide mass is situated. This change was probably not related to the east-west fracturing and may have been important in localizing the mineralization. In that case, the east-west break may not be entirely pre-mineral but may correspond to the dike fractures that were formed before the final stages of the mineralization period. There has also been some post-Paleozoic movement along this break.

Copper Chief.—The original sulfide mass of the Copper Chief-Iron King deposit was formed at the contact of the granitic porphyry which intruded rather siliceous and not very schistose greenstones. Schistose portions of the fine-grained margin of the granitic intrusive furnished the material chiefly replaced.

The outcrop of the deposit is much smaller in area than the greatest cross-section of the sulfide mass and, similarly, the mass is shown to pinch to almost nothing in the lowest mine workings. Thus, it appears that the Copper Chief deposit represents an exceptional swell in a very small channel of mineralization.

Genetic Relations

The localization of the most important mineralization of the district by an inverted pitching trough rather than a normal one is strong evidence that the mineralizing solutions came from below rather than from above. Furthermore, the similarity of this mineralization to others known to be the work of juvenile or magmatic solutions places the burden of proof upon any one who would assign to them a different origin. Therefore the source of the mineralizing solutions is to be sought in an igneous magma.

The United Verde diorite seems to be most closely related in time and space with the two large ore deposits of the district. However, there does not appear to be any United Verde diorite near the Copper Chief or similar diorite related to a number of pyrite-chalcopyrite schist-

replacement copper deposits farther south, in the Bradshaw quadrangle, which are believed to have a common origin with the schist-replacement deposits of the Jerome district. The Bradshaw granite best meets the requirements on this basis. The facts outlined make it seem reasonable to consider the Bradshaw granite and the pre-Cambrian intrusives younger than the Bradshaw granite as all related to a single period of batholitic intrusion and regional dynamic metamorphism. Looking upon the United Verde diorite as a late phase of the Bradshaw granite, the logical conclusion is that the mineralizing solutions that formed the deposits of the Jerome district and the other schist-replacement copper deposits of the Yavapai schist belt originated in the magma reservoirs of the Bradshaw granite. This explains the apparent preference of the jaspery quartz masses for quartz porphyry as well as diorite contacts, and perhaps also numerous minor showings of mineralization which seem to be related to quartz-porphyry contacts. In this latter case, however, the relation may be entirely a structural one.

Important ore deposits are usually related to apophyses or outlying porphyritic phases of batholitic intrusions rather than along the walls of the main masses. Thus, if the above relation to the Bradshaw granite is correct, the Jerome district is more favorably located for ore deposits than the area farther south in the central portion of the batholith.

Prospecting in the Jerome District

It is believed that with the present knowledge of surface indications underground exploration is, in general, only justified by promising surface showings. Where the surface of the ore-bearing formations is covered with material younger than the mineralization, in a district where important ore deposits have already been found, underground exploration is sometimes justified in the absence of such indications. The assumption that this is true in the Jerome district is responsible for much of the activity of the prospecting companies.

The experience gained by prospecting and geologic study has established certain points of importance in connection with further prospecting. In exploring the pre-Cambrian formation beneath the Paleozoic cover, the character of the rocks encountered may be the chief factor in guiding the work. Large masses of well developed schist are essential to the occurrence of important orebodies of the schist-replacement type. The nearly black, chloritic schist and the light-colored, sericitic, quartz-porphyry schist are the most favorable varieties. Contacts between the schist and a relatively massive and impervious formation are favorable localities, especially where the impervious material forms a hanging wall to the schist. Reëntrant angles or embayments in such a contact are most favorable. Possibly where the impervious hanging wall consists of a

mass of United Verde diorite and where the andesite dikes are abundant, conditions are more favorable than otherwise, but this is open to question. The foregoing considerations also apply to prospecting where the pre-Cambrian formations are exposed at the surface, but in this case have relatively little weight in the absence of extensively developed gossan material.

Iron oxide weathered from the highly ferruginous Cambrian sandstone and deposited in the underlying formation has sometimes been mistaken for gossan material. Similarly the iron-staining developed by the weathering of the more chloritic rocks of the Jerome district has been incorrectly referred to as evidence of mineralization. Much unprofitable work has been done exploring the areas beneath surface exposures of jaspery quartz. Where there is no gossan or other evidence of important sulfide mineralization, these quartz masses do not warrant exploration.

Tertiary "Malpais" dikes, as they are called, have been found in many of the mine workings of the Jerome district. These dikes are dark brown to black in color and are often porphyritic or amygdaloidal; while the small pre-Cambrian dikes are nearly all light gray to green in color. Crustified quartz veins and silicification of the wall rocks associated with the Tertiary dikes have sometimes been confused with the pre-Cambrian mineralization.

The writer wishes to acknowledge indebtedness to H. V. Winchell for helpful suggestions and discussion, and to the officials of the United Verde Copper Co. for the liberal policy pursued in connection with the geologic work.

Geology of Pachuca and El Oro, Mexico

By Horace V. Winchell, Minneapolis, Minn.

(New York Meeting, February, 1921)]

An examination of the Pachuca[1] and El Oro districts in July, 1920, although cursory and incomplete, disclosed facts of more than passing interest to the student of ore deposits; and inasmuch as the literature on these districts is limited and some of their features are little understood, a brief account may prove valuable as reconnaissance observations to be amplified by more thorough and competent observers in the future. Where no complete monograph of important mining camps has been published a partial presentation of the data is often useful, as it sometimes leads to fuller and more accurate descriptions, by stimulating discussion.

The following notes are presented with the consent of A. F. Main, managing director of the El Oro Mining and Railway Co., whose fund of information and wide acquaintance with both districts alone made it possible to cover so much ground in a short time. I have also to acknowledge gratefully the aid of C. A. Lantz and D. S. Calland of the Santa Gertrudis and Real del Monte companies, respectively, in connection with permission to inspect geological features of interest in their mines.

Pachuca, Hidalgo, Mex.

This mining camp has been described by Ezequiel Ordoñez[2] and by A. F. J. Bordeaux.[3] Reported to have been discovered in 1522, it has produced silver valued at between $250,000,000 and $375,000,000 Subsequent to the discovery here, by B. de Medina, of the patio process, in 1557, its production was increased, and now, using cyanidation, the output of the camp is nearly one-sixth of the production of the world.

Topography and Climate

The Pachuca district consists of a series of mountains and valleys, the minor features of which extend in an easterly and westerly direction and reach elevations of 8000 to 10,000 ft. (2438 to 3048 m.) above the

[1] A map of this district has been placed in the Engineering Societies Library.

[2] Mining District of Pachuca, Mexico. *Trans.* (1902) **32**, 224.

[3] Silver Mines of Mexico. *Trans.* (1908) **39**, 360.

sea. The range as a whole constitutes a continental divide; its axis runs northerly and southerly and separates the waters that flow westerly into the Pacific Ocean from those flowing easterly into the Gulf of Mexico. On the north and east, the surface falls rapidly down to the *tierra caliente* and is cut by deep cañons. In this warm country, about 4000 ft. (1219.2 m.) below the summit is a cañon 2500 or 3000 ft. (762 or 914 m.) deeper that drains the region on the eastern slopes; here are raised coffee and such tropical fruits as bananas. Dense forests, heavy undergrowth, and much soil cover the land. Cloudy weather prevails over a large part of the year; the abundant rains do not run quickly off but are held in the soil and in the vegetation that conceal the underlying rocks.

On the western slope we are almost immediately in different atmosphere and country; the vegetation is relatively sparse and the conditions semi-arid. The rocks are either bare or covered with laterite; they are often weathered and rotted to a considerable depth and the eye has an unobstructed view over large areas. The sky is clear, the sun beats hotly down, and although there are frequent showers in the afternoon, sharply limited to portions of the area, the waters either run off quickly or are rapidly absorbed by the rotted rocks.

According to the best weather data obtainable, there is a marked difference in the amount of rainfall on the two slopes, within the few miles occupied by the operating mines. At Pachuca, the average rainfall, for the years 1909 to 1912, inclusive, is given by Mr. Lantz as 11 in. per annum; according to Mr. Calland, the records of the Real del Monte Co. show at Loreto, where the Pachuca mill is situated, a precipitation of 21.22 in. (53.97 cm.); at the village of Real del Monte, 32.76 in. (83.2 cm.); and at Velasco, about 3 mi. (4.8 km.) north of the Guerrero mill of the Real del Monte, 59.33 in. (150.7 cm.). These facts are mentioned because they suggest a difference in the chemical activity of surface waters, which as will be explained later seem to have played an important part in the genesis of these extensive ore concentrations.

Geology

We have to do here with Tertiary eruptives similar to those that contain so many of the mines of Mexico and the southwestern part of the United States. The underlying basement on which these volcanics were deposited is probably Cretaceous sediments, but none were observed in the area covered by these notes.

The oldest rock seen is the older andesite. This is a thick flow of inconstant texture and varies from a massive rock with few phenocrysts to a highly porphyritic mass in which are many whitened feldspars and some altered pyroxenes with occasional biotites. Toward the north and west this typical andesite is succeeded by a quartz andesite, or dacite,

much more siliceous, lighter in color and with many quartz phenocrysts ranging in size from 1/32 to 1/8 in. (0.79 to 3.175 mm.) in diameter. Whether the dacite is a phase of the older andesite or a separate flow was not ascertained. No definite line of contact was seen. Both of these rocks contain fragments of darker fine-grained andesite and both at times exhibit marked flow structure. They can hardly have been intrusives, for there is no remnant of any covering rock. Moreover, at times the vesicular or amygdaloidal texture characteristic of rocks cooled near the surface is found.

The older andesite appears on the surface over the greater portion of the area thus far developed by mining operations. It can be traced continuously from a point more than 1 mi. (1.6 km.) east of the Guerrero mill of the Real del Monte Co. to a point west of the Bordo shaft, where it passes beneath later eruptives, about 12 mi. Its width on the surface is much less, since its northern margin is south of the northern limit of underground mining operations and may be seen just north of the valley in which are situated the Paraiso and Santo Tomas shafts. Its southern limit of exposure was not traced.

Within this older andesite are east-west zones of extensive and profound thermal alteration in which the rocks have been altered over areas probably several miles long and often hundreds of feet wide. The altered rock is softer and lighter in color than the fresh andesite and carries occasional pyrite crystals, with calcite films and veinlets, and even quartz incrustations. It is in these altered zones in the older andesite (and dacite) that the important ore deposits have been found and developed.

The succeeding rock is later andesite. This consists of flows, breccias and tuffs, all highly porphyritic and generally less massive than the older andesite. It is, however, sometimes thick-bedded and dense and so similar in appearance as to be distinguished with difficulty except on weathered surfaces. It weathers in various colors from reddish brown to bluish or purplish gray and covers large areas north of the actively mined district. In places, only a thin shell of this rock is left; elsewhere shafts go down through it into the lower andesite. In the Girault tunnel, the contact is well exposed; near the Trompillo shaft there is a steep fault contact between the two formations with the Vizcaina vein lying between. This later andesite has not been considered promising ground for exploration and has produced but little ore. Whether this prejudice is based on its actual poverty or not is a question that can be answered only by future and deeper development, where this later formation attains great thickness. So far as I could learn, the only workings in it are comparatively shallow (at least in the central portion of the camp) and do not attain depths where, even in the older andesite, valuable orebodies are first encountered.

Still younger, and overlying the later andesite, is rhyolite. This rock is cream colored and occurs on the surface in thin flows and tuffs, often much weathered and cut by small arroyos. Rhyolite or quartz porphyry dikes of the same material are also seen on the surface and underground throughout the district from the Santa Gertrudis north to the Capula and from the crest of the main divide above Real del Monte west to and beyond the Bordo. These dikes have various strikes and dips but the larger and more continuous ones strike and dip parallel to the main productive vein systems. On the surface and underground these rhyolite flows and quartz porphyry dikes are cut by the vein fissures, which are sometimes particularly well mineralized and contain large stopes where they abut against or rest upon the dikes. At Capula the vein, carrying ore in two or more strands, is in a quartz porphyry dike of great longitudinal extent and from 50 to 100 ft. thick dipping southerly about 50°.

Cutting later andesite, but of undetermined age relation to the rhyolite, is basalt. This rock is seen in east-west dikes from 5 to 40 ft. (1.5 to 12.2 m.) thick and a general northerly dip in the country west of the Pachuca mineralized area. It is not much weathered. It stands up in bold outcrops and is dense and tough and contains abundant olivine phenocrysts. The later andesite near these dikes has been weathered and oxidized, and the red belts have sometimes been mistaken for veins.

Vein Systems

So far as could be observed, all the veins of the Pachuca camp belong to the same general period of fracturing. Although some veins terminate against or are faulted by others, the evidence tends to show that all of the productive veins thus far developed were formed and primarily mineralized at one period, and from the same agencies and structural disturbances. Nevertheless there are two systems of veins; one striking east and west and one striking north and south. The east-west veins are said to have been first discovered and worked, although the north-south veins, which are rich and wonderfully productive, have also been worked for many years. The latter occur in the eastern part of the camp, largely in the property of the Real del Monte Co. Most of the east-west veins have numerous branches extending both northeasterly and northwesterly, forming what has been termed "linked vein" structure, quite typical of rock fracturing on a large scale. Some of the branches extend throughout the intervening space between the stronger east-west veins; others apparently die out before reaching any great distance. The full extent of the north-south vein system has not yet been disclosed, nor is it known how many such veins remain to be discovered. They seem to terminate on the south against the great Vizcaina fissure, although the Dios te

Guie is said to be displaced by the Vizcaina and to extend a short distance south of it, without however containing much ore.

The principal veins of the east-west system are the Santa Gertrudis, Fresnillo, Analcos, Vizcaina, Maravillas, Santa Ana, and Polo Norte. La Corteza and El Lobo veins strike northwesterly, while the Veta del Tajo, Cristobal Colon and Florencia strike to the northeast.

So far as developed, the north-south veins, in their order from west to east, are the Dios te Guie, San Sabas, Purisima, Santa Ines, Santa Brigida, and Veta de la Reina. The stoping width of these veins varies from 3 to 40 ft., and some of the oreshoots, such as that in the Purisima (N-S) vein are 400 m. long. The large east-west veins contain some of the longest oreshoots ever developed, reaching 1000 m. in the Santa Gertrudis vein and as much in the Vizcaina. The horizontal dimensions of the oreshoots generally exceed the dip length. The bearing of this fact on the question of ore genesis will be suggested later. The dip of the east-west veins is generally to the south, at angles of from 65° to 80°. Local north dips are not uncommon. The north-south veins dip both easterly and westerly; it is said that the former are the more productive.

The veins occupy planes of fracture and zones of shearing and are composed of crushed country rock more or less completely replaced by quartz and other vein minerals. The best veins, from the standpoint of productivity, are in and course through the zones of greatest rock alteration, and yet there was occasionally observed a belt or barrier of fresher looking rock just before coming to a vein. Such a belt lies just east of the east-dipping, north-south Purisima vein.

There are evidences of two generations of quartz deposition. The first-period quartz, which replaced the broken andesite and often presents a banded or curved structure suggestive of its deposition around rock fragments, is white and bony or ivory-like. Upon and around this white quartz and in its interstices is a darker, later quartz carrying silver minerals with pyrite and a little galena and, still rarer, crystals of blende and chalcopyrite. Where not replaced by quartz, the vein filling is sericitic or kaolinic. The quartz is usually broken into fragments, especially where the veins are wide, but is sometimes solid and "frozen" to the hard country rock. Some veins have good walls; some are in "bad ground;" some are accompanied by well defined planes of movement on one or both walls or in the veins themselves. The best and most abundant ore seems to be in those veins that are quite open to the passage of ground water. There is not a great flow of water, considering the length of underground development. As the rainfall at Pachuca is so much less than that at Real del Monte, it is probable the eastern mines make more water than those in the western camp.

It is a singular and significant fact that on the western slope oxidation extends to the lowest productive levels, about 2000 ft. (610 m.) from

the surface. This is not intended to imply that no sulfide minerals remain. The sulfide of silver, argentite, is the principal ore mineral, and pyrite is quite abundant; but the quartz is honeycombed, rhodonite and rhodochrosite are more or less altered to psilomelane and pyrolusite, horn silver is apparent and calcite coatings are found on the joints and in cavities. The effect of weathering is naturally greater as one approaches the surface, and indeed, not only the silver but the quartz itself seems to have been dissolved out of the upper portion of the veins and carried downwards. It is only occasionally that quartz and ore persist upwards to the grass roots. Few stopes extend high enough to make their presence known by surface settling, and the average prospector acquainted with the camp states without hesitation that good ore can hardly be expected short of 400 or 500 ft. (120 or 150 m.) in depth. This has a familiar sound, but seems to have more than an element of truth here. Long oreshoots, already fully explored and mined upwards to their terminations sometimes have tongues projecting to the daylight; but the average line of the upper margin of the long shoots is perhaps 150 m. from the surface. Indeed, there seems to be a general parallelism between the surface topography and the configuration of the oreshoots below. This rule may possibly not hold where there is a considerable thickness of recent volcanics, even though such rocks are cut by the vein fissures. It must be admitted that there is no rule without exceptions and that no general statement is applicable to every vein in the district. There are stopes that reach the surface; there are oreshoots that do not persist downwards; there are veins whose production has come from within 500 ft. of the present surface and others that are chiefly barren to that depth and productive below it. Nevertheless there is a remarkably persistent general relation between the surface and the upper limit of the pay ore.

This relation becomes still more striking when we consider also the termination of the ore downwards. There seems to be little doubt in the minds of those familiar with Pachuca that the mineralization has a rather abrupt and very definite termination in several fully explored mines, and that the productive area of the veins is confined within a vertical range of about 2000 ft. (610 m.). Moreover, in no one oreshoot is the full extent of this zone mineralized. The average height of the pay-ore zone is perhaps 1500 ft. (457 m.), in many cases less and in a few cases more. My knowledge of the conditions that prevail at the bottom is largely gained from others. It is common knowledge, however, that the veins scatter, the quartz diminishes, the values fall off rapidly and only occasionally are found any sulfide or base minerals such as blende and galena. This general situation is not in any way exceptional for veins in Tertiary eruptives, but the explanation may not always be the same nor the geological record so easily read.

Origin of Ore

In considering the possibilities of a mining district an understanding of its geology and something as to the probable genesis of its ores is valuable as a guide in explorations and in appraising its future. The facts given may be summarized as follows:

1. The country rock is a series of Tertiary volcanics.

2. The vein fissures cut the entire series from the oldest to the youngest.

3. Parallel to the veins are quartz porphyry dikes of considerable extent. These are not universally known, but are in sufficiently constant association with the veins to be taken into consideration as possible agents in ore genesis.

4. The country rock is not only widely sheared and fissured but presents evidence of profound alteration by thermal waters over wide zones.

5. The country rock is still further altered by weathering from the surface down to considerable depth.

6. The veins are largely quartz, but this quartz seldom comes to the surface and never in such quantity as in the veins underground. This statement is true of veins and their outcrops whether found on the crests of mountains or in deep valleys. In other words, both quartz and ore lie for the most part some distance below the surface.

7. Oxidation and leaching persist to the lowest levels of silver enrichment. Horn silver, native silver, and argentite (the latter greatly predominating) are the ore minerals. They occur in white quartz, which is itself secondary, or as incrustations or cavity fillings together with pyrite and an occasional speck of chalcopyrite and galena. For the most part, oxides of iron and manganese are present though in diminishing amount as depth is gained.

8. There are seldom any massive sulfide orebodies in the veins beneath the oxidized ore, nor are there in the upper levels the large masses of oxidized material that often indicate the former presence of heavy sulfide bodies.

9. The major axes of the oreshoots are more nearly horizontal than vertical; so much so that the Pachuca camp is almost unique in this respect.

10. The vein quartz dies out downwards, and even large veins dwindle into a series of scattered and unmineralized stringers.

11. All the mineralized veins belong to one general period and contain the same kind of ore. The products of the camp are silver and gold in the ratio of about 5 gm. of gold to 1 kg. of silver.

With reference now to the richly productive portion of the Pachuca camp, the history of ore formation may well have been somewhat as follows:

Having accumulated in large mass, the andesites, both older and later, slowly cooled, and zones of shearing were produced by shrinkage and subsidence. Through the multitude of cracks and fissures, vapors and hot waters penetrated the rock and effected widespread alteration, depositing at the same time barren pyrite in disseminated crystals over zones of considerable width. When these fissures extended to sufficient depth there was another outburst of magma, this time quartz porphyry and rhyolite, followed by the usual period of hot-spring activity. At this time the first quartz was deposited in and along the fissure already formed and replacing and silicifying the broken andesite along the shear zones. The period of subsidence and fracturing was not yet at an end, for the quartz porphyry itself, after cooling sufficiently, was fractured and fissured and somewhat mineralized.

Then, with the dying down of fumarolic and hot-spring activity, came the opportunity for surface waters, which up to this time had been operating solely on the exposed surfaces, to begin working their way downward along the fissures and shear zones, oxidizing and dissolving the scattered sulfides and carrying them to new resting places at lower levels. It is not known how much erosion has taken place since these Tertiary rocks were formed, but that it may well have been thousands of feet is shown by the depth of the valleys in the immediate vicinity.

On the western slope of the mountains chemical changes are rapid and with an already altered rock on which to operate the upper parts of the veins were constantly and successively oxidized, leached of quartz, silver minerals, pyrite and gangue minerals while the surface was eroded and carried away. Always the values were held in the veins and carried downward in advance of the dissipating forces of erosion, and times without number the little films of argentite that had been deposited at a safe depth and were becoming endangered by the slow approach of the surface, were removed still deeper. In this way we have the cumulative result, first, of ages of primary deposition during which perhaps no commercial orebodies were formed, and, second, of a long period of weathering under a hot sun and climatic conditions distinctly favorable to secondary enrichment.

The proof of the theory is in the character of the minerals, the leaching and other evidences of the work of descending waters, as well as the shape of the oreshoots, their correspondence with the topography of the surface, the paucity of quartz at grass roots and its diminuendo habit beneath the ore accumulations. In short, all the broad phenomena of the district seem to be in accord with this theory and with no other. It is supported by both the positive proof and the negative facts, by the minerals found as well as by those that are not present; it is a most excellent example of the formation of large and deep orebodies by secondary sulfide enrichment. But there are other features of interest.

Climatic Effects

Like forces working on similar materials under similar conditions, for an equal length of time, may be expected to produce similar results. Thus, with equal precipitation and evaporation over the entire district, uniform surface gradients on both sides of the divide and similar rocks uniformly sheared and fissured so as to offer equal receptivity to drainage, there would probably result a similarity in topography and in the subterranean products of weathering agencies. Where the materials are similar but the conditions are known to vary, it is reasonable to attribute heterogeneity of products to such variance of conditions; and where the rocks are different, different products may be formed by the operation of similar forces under similar conditions. At Pachuca, the rocks are, in general, similar mineralogically and structurally; the operating forces are similar, although not of equal intensity nor volume; the time factor is practically constant; but the products have a wide variance in different portions of the district.

Reference has been made to the marked difference in the annual precipitation on the western and eastern slopes of the continental divide, but its effect has not been fully described. The first result to be noticed is in the different sculpturing of the surface. On the eastern slope the topography is rougher, the slope gradients are steeper, the changes in elevation more frequent and abrupt. On the western slope there are many long, smooth, gently sloping hillsides and but comparatively few sharp and deep cañons. As a consequence, the rainfall, if equal over the two areas, would run off faster on the eastern than on the western slope, and a smaller amount would percolate downward into the rocks. But with the much greater rainfall on the eastern slope, it might be expected that the effect of surface waters would be at least as great, and oxidation as deep in Real del Monte as in Pachuca. In fact, considering alone the much greater precipitation, one would expect deeper and more intense oxidation. But the reverse is the case. Sulfide minerals are rarely found within 300 ft. (91 m.) of the surface in the western part of the camp, while they are abundant within 70 ft. of the surface about 1 mi. (1.6 km.) east of the Guerrero mill, on the eastern side of the range. It is probable, as already stated, that there is more water in the mines and in the ground generally, on the eastern than on the western slope. Why then is there such a marked difference in the depth to which oxidation has extended?

No doubt many factors enter into the problem. It is evident at a glance that erosion is more rapid, and that it nearly keeps pace with oxidation, on the eastern slope. But there are other reasons why sulfides in that section are so much nearer the surface. It may be explained in part by the fact that the rain waters are not so active chemically after soaking through the soil. There is perhaps not enough difference

in temperature to make any material difference, although such difference probably exists. It is perhaps more largely due to the fact that the rank vegetation that covers the eastern foot hills deprives the rain of its oxygen and hence the underground waters in that section are comparatively inert. In this respect the Pachuca camp offers a rare example of the effect of differing climatic conditions on the depth and character of mineralization in veins.

In another respect, also, it is interesting. Many observers here and in other districts have noted the fact that there is oxidation below the present water table and have attributed the phenomenon to a change of water level in comparatively recent time. This does not appear to be the only, nor indeed always the more probable, explanation. Where, as is probably the case here, the entire volume of ground water is slowly moving downward, and yet is ever renewed by annual rainfall, there must be oxidizing action until all the oxygen is consumed, and thus, even below the surface of the subterranean water table, extending downward perhaps several hundred feet, the sulfide minerals will become oxidized. It is only stagnant water or water that has performed its work and become exhausted, that is inert. Where there is an outlet at some greater depth and the waters are descending, the work of oxidation may and frequently does proceed far beneath its apparently stationary level.

El Oro, State of Mexico

For his data on El Oro, the writer is indebted not alone to his personal examination but to a report on the camp by Waldemar Lindgren, written in 1913. From this report are taken the following more general statements, in order to lay the foundation for points that seem to be of special interest to the economic geologist:

"The district of El Oro is situated on the high plateau of Mexico, near its western edge, at an elevation of about 10,000 ft. (3048 m.). On this part of the plateau broad valleys are separated by irregular groups of mountains rising 2000 to 3000 ft. above the depressions. The valleys are filled with volcanic tuff and detritus. The mountains are largely built up of volcanic flows, mainly andesite, but at many places the underlying older rocks are exposed. The latter consist of calcareous shales of Cretaceous or Jurassic age and in places contain an older series of igneous rocks intruded into the shales and exposed by erosion.

"The geological sequence is then as follows:

1. Calcareous shale with some sandstone and limestone.
2. Older Tertiary igneous rocks intruded into or poured out on these sedimentary shales.
3. Formation of fissure veins intersecting shales and older igneous rocks.

4. Epoch of erosion.

5. Late Tertiary and recent igneous rocks, chiefly flows of lava and tuffs, resting on eroded shales, older igneous rocks and veins and showing no mineralization.

6. Recent epoch of erosion.

The Formations

"*Later Andesite.*—The younger surface lavas are mostly massive, dark gray hornblende-andesites which are oxidized and disintegrated near the surface, but show no mineralization, nor do they contain pyrite. Toward the valley agglomerates and tuffs gradually take the place of the massive rocks. Where the contact with the shales is exposed by mining operations, a few feet of reddish stratified material of fragmental origin often rest directly on the shale.

"The thickness of this lava is manifestly affected by the recent erosion. Along the San Rafael lode, south of North shaft, it is less than 200 ft. (60.9 m.), but north of this point it increases to 400 ft. (121.9 m.) and at Tiro Hondo and San Patricio shafts it is about 600 ft. Under the summit of the hill the thickness is 1000 ft. Dikes and intrusive necks of this lava are found in the adjoining Esperanza mine.

"*The Older Andesite.*—The older andesite is a greenish dense rock which has been greatly altered by the vein-forming agencies and now contains much pyrite, calcite and sericite. It occurs as thick intrusive sheets, or as irregular masses in the black shales, also as smaller dikes. On the property of the El Oro Co. none of this rock reaches the present surface or the old surface underneath the younger andesite, but it forms a thick flat body which is about 600 ft. thick; it was first encountered in the northern part of the property along the San Rafael vein about 600 ft. below the capping. Farther south it lies deeper, being near the Interior shaft at about 900 ft. below the capping; and in the Carmen mine its top lies again about 800 ft. below that surface. The lower contact with the shales has been found in the northern part of the mine. Throughout, this andesite sheet, or sill, is faulted by the fissure of the San Rafael, the vertical throw being, in the vicinity of the Somera shaft, about 670 feet.

"Down to about the 1300-ft. level, the andesite appears only in the foot wall. Below the 1300-ft. level, it begins to appear in the hanging wall, and continues to form that wall down to the 1600-ft. level, the lowest point reached." (Since Doctor Lindgren's examination the mine workings have developed ground far beneath the lower margin of this sill).

"The older series of andesite is represented both by intrusive rocks and lava flows.

"*The Sedimentary Rocks.*—The predominating rock is a black, bitu-

minous shale, well stratified and often containing much calcite, in fact grading into a calcareous shale and occasionally into a black granular limestone. In places the shale contains embedded masses of a dark gray friable sandstone. This sandstone is more abundant in the deep levels, and is typically present at the station and crosscut of the Somera shaft on the 1300-ft. level. When examined in the field this rock was held to be of tuffaceous origin, but the microscope has shown it to be a pure quartzose sandstone.

' The sedimentary rocks lie horizontal, or at slight dips that exhibit no marked irregularity. A total thickness of 1300 ft. of strata is exposed in the workings."

The Veins

We have at El Oro a series of veins bearing gold and silver minerals in a quartz-calcite gangue. These veins are fissures that were filled by replacement and infiltration. Some of them are evidently fissures of considerable displacement. They cut through the black shales and through the sill of andesite that was intruded horizontally into these shales, and which, as a natural consequence, is both overlain and underlain by the shale beds. After the veins were formed and mineralized, they were subjected to the action of surface waters for a considerable time, and an unknown extent of their upper portions was removed by erosion. They are oxidized to the depth of nearly 1000 ft., and their silver content at least was secondarily enriched by the action of descending waters. The general strike of the veins is about north 30° west, and their dip at varying angles to the west. They have branches in both foot and hanging walls, but more numerously in the latter.

After a long period of weathering and erosion the country was covered by more recent lava flows, which were in turn weathered and eroded until in some localities the underlying shales and one vein, the first discovered Descubridora, are exposed on the surface. These lava flows and volcanic tuffs are not penetrated by the veins which cut the underlying rocks.

The veins are, in turn, cut and displaced by north-dipping east-west faults. These faults are unmineralized, except by a little calcite, which may be of recent deposition. Their general effect has been to step the country down to the north. The direction of movement along several of these faults has been diagonally downwards to the east.

The andesite sill varies in thickness from 400 to possibly more than 700 ft.; it has been cut and displaced by the vein fissures with throws of several hundred feet.

The most productive orebodies have been found in the veins where they lie within the shale overlying the andesite sill and adjacent to it where, by reason of faulting movement along the vein fissures, the shale is brought in opposition to the andesite. There is also some ore found

where the veins lie wholly in andesite, but not in large quantity anywhere above the lower margin of the lower faulted segment of the sill. Developments indicate that the veins are not enriched for any distance below the andesite. Quartz indeed is found to persist for some distance beneath it, but with diminishing tendency downward, and with smaller oreshoots.

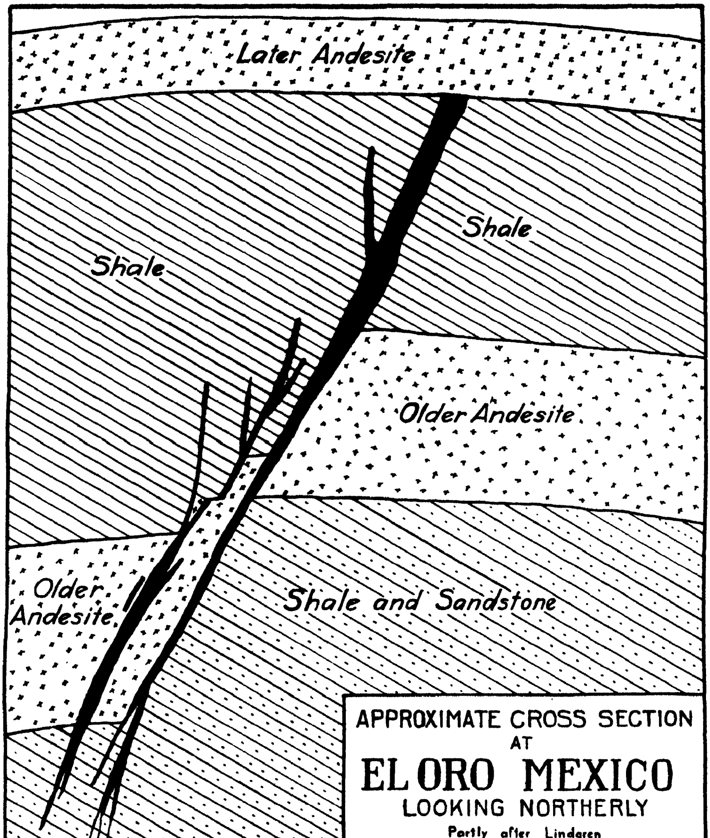


FIG. 1.

The quartz scatters in stringers of diminishing strength and dies out. The lower stretches of the veins contain some of the baser sulfides, such as blende and galena.

The commercial orebodies occur in long relatively horizontal oreshoots better mineralized in the upper levels. Some of these shoots have been definitely bottomed while others are still being pursued through their downward reaching lobes and tails.

Origin of the Ore

It is seldom that the facts observable suffice to point strongly to a particular mass of eruptive rock as the source of vein mineralization. In this respect El Oro is of particular interest. The position of the ore-bodies in the ground and their relation to the andesite sill, the dwindling of the quartz and mineralization downward suggest that this was the fountain head of ore deposition. There is nothing to suggest the presence of other sills or masses of intrusive rock and many facts that lead one to doubt their existence. This holds true of all the veins thus far developed. They are large and rich above the sill and poor and small or entirely pinched out beneath it. This could hardly be the case if they had been formed from solutions rising from greater depth. Further careful study should prove or disprove the integrity of this theory. It is at least useful as a working hypothesis for it not only suggests the futility of costly exploration beneath the sill but at once shows the attractiveness of territory overlying the sill still unexplored and now covered by later andesite, which caps the vein-carrying shale formation.

Value of Geological Study

In these two camps, we thus find geological data that encourage further exploration and indicate where it is most likely to be successful, while in many camps are found facts that, properly interpreted demonstrate the utter folly of large expenditures in the search for orebodies, the existence of which is extremely problematical. Such explorations are now in progress in some of the older western mining camps and hundreds of thousands of dollars are being expended in utter defiance of the easily read geological history of the districts wherein such work is being prosecuted.

DISCUSSION

FREDERICK MACCOY,* Mexico City, Mex. (written discussion).—There have been several instances in the history of the mines on the Vizcaina vein, when water was quite a problem. To the north of this vein there are zones of andesitic breccia, which retained large quantities of water. One of these zones was cut, in December, 1895, by a crosscut in the Camelia mine, on the 290-meter level. The flow reached 3600 li. per min., making a total flow in the San Rafael, Camelia, and Sotol mines of 5000 li. per min. As the capacity of the pumping equipment was about 3000 li. per min., the water rose rapidly until June, 1896, when the additional pumping equipment was able to control the flow. Constant pumping for three years was required, in the San Rafael, before the

* Director General, Negociacion Minera de San Rafael y Anexas, S. A.

bottom levels were reached; then it was found that these levels in greater part had to be re-opened, in many places counter driving at some distance from old workings.

In 1906, another body of water was cut, but bulkheads had been constructed in anticipation of such an occurrence.

The San Rafael mine is driving exploration crosscuts to the north of the known district, and because of the possibility of encountering other reservoirs of water, maintains a pumping equipment sufficiently large to take care of 6800 li. per min. The present flow in San Rafael is 2000 li. per min., all coming from a drift 450 m. long.

ROBERT HAWXHURST, JR., San Francisco, Cal. (written discussion).—The first part of the paper is a timely reminder that to secondary enrichment is due the economic value of orebodies in many gold- and silver-bearing veins that occur in Tertiary eruptives, and hence, when such conditions prevail, of the futility of searching for profitable ore below the zone of alteration.

I cannot agree with Mr. Winchell that rain waters are rendered inert by contact with vegetation, however rank. On the contrary, the capacity of such waters for solvent action is thereby increased through the absorption of additional carbonic acid, which is supplied by decaying vegetal matter. Evidence of this is found in the familiar "sour" soil, of newly cleared land in humid climates, which requires an alkali treatment to fit it for cultivation. The difference in depth of the oxidized zones, at Real del Monte and Pachuca, is fully explained by the higher water table and the increased rapidity of erosion, in the district of greater rainfall.

General Geology of Catorce Mining District

By Charles Laurence Baker, Berkeley, Calif.

(Wilkes-Barre Meeting, September, 1921)

The district of Catorce, San Luis Potosi, ranks among the first half-dozen silver-producing camps of Mexico. Mining has been more or less continuous there for 150 years. The large producing mines, Mapimi, Mazapil, Palomas, Musquiz, Minas Viejas, now for the most part idle or abandoned, are grouped around the head of two valleys: one, which drains to the west, falls 2000 ft. (610 m.) in 4½ mi. (7.2 km.) between the main Catorce and the western foot of the range and is characterized by steep cañon walls; the other, east of the main town of Catorce from which it is separated by a high divide, heads in a number of steep tributaries above the basin in which lies the town of Potrero and flows nearly north for 10 mi. (16 km.) to the northern base of the range; below Potrero, it has for the most part gently sloping valley walls and a broad valley floor.

The Sierra Catorce is the highest range of the region, the summits reaching altitudes of over 11,000 ft. With the exception of a few cliffs of the erosion escarpment of the lower limestone and the cañon walls below the main town of Catorce, which is the highest town in Mexico, the gently rounded slopes are characteristic of a maturely eroded mountain range composed of folded sedimentary rocks. As is usual in desert regions, the heavy-bedded limestones are the most resistant to erosive forces. Folded into anticlines, they form the mountain ranges; and the intermontane valleys and basins, originally carved out of less resistant shales, clays, marls and sandstones, are covered by debris forming a mantle with surface gradually rising upwards toward the hills. In time, the debris buries, in deposits formed from their own ruins, all but the highest summits of the original mountains; some of this region has already nearly reached this stage. Most of the debris is firmly cemented by calcium carbonate (caliche). Solution of the limestone by meteoric waters afforded the supply of lime carbonate, which has been redeposited upon evaporation of the waters as a cement binding the boulders, pebbles, and finer detritus.

According to tradition, the higher portions of the Sierra Catorce were forested with pines and cedars but none remain within the mining district, although cedar is found farther south in the range. The vegeta-

tion consists of grasses, in the higher and more humid places, and of bush and scrub, in the lowlands and on the steeper slopes. The characteristic plants are yuccas and other agaves, cacti, acacias, and creosote bushes (*gobernadora*). In vegetation, topography, prevailing hues of the desert landscape, the Catorce region greatly resembles parts of Nevada and the Mohave Desert.

Previous Investigations

The earliest geologic study of Catorce appears to have been made by Joseph Burkart, who visited the camp about 1830. In his book[1] he gives a cross-section that is surprisingly accurate in its lithologic observations. Humboldt mentions Catorce but never saw it. The first discovery of Jurassic strata at Catorce was made by the Russian paleontologist, Nikitin. Castillo and Aguilera[2] described some fossils and give some notes on the sedimentary rocks. S. J. Lewis[3] has recently published some notes on the ore deposits of Catorce.

Metamorphic Rocks

Metamorphic rocks are exposed in the cañon from the main town of Catorce westward to near the cañon's mouth. They may also be exposed in the cañon farther south, which extends southwest from Alamitos Diaz and Mantanzas. The metamorphics are divided into two series; the relationships between the two are unknown. The probably older series outcrops between the Purisima tunnel and the main town of Catorce. It consists mainly of sericitic and talcose schists (green in color with fairly well defined schistosity), greenish rocks (probably originally porphyries), and some thin layers with the appearance of andesitic tuff breccias. A less metamorphosed series probably forms a steeply dipping anticline with axis in the vicinity of the lower town of Catorce and outcrops between the Purisima tunnel and the exposure of the unmetamorphosed sedimentaries near the mouth of the cañon. These rocks are mainly green or purple arenaceous or shaly rocks (possibly tuffaceous at least in part), weathering in splinter-like forms, and interbedded with rusty-brown conglomeratic quartzites having rounded quartz pebbles and a matrix of clear quartz. This series was originally partly or entirely sedimentary. Metamorphism of this series is by no means extreme, although incipient slaty cleavage or schistosity is devel-

[1] "Aufenthalt und Reisen in Mexico in den Jahren 1825 bis 1834." Stuttgart, 1836.

[2] Fauna Fosil de la Sierra de Catorce. *Boletin* 1, Instituto Geologico de Mexico, (1895).

[3] Ore Deposits of Mexico. *Min. & Sci. Pr.* (June 26, 1920) **120**, 938.

oped in the less compact rocks. Both these series are injected with veins, stringers, and lenses of quartz.

The age of the metamorphics is not known; they might in part be as old as pre-Cambrian or as young as Jurassic. Rocks very similar to both series have been described by Dr. Carl Burckhardt, as the lowest exposed at Zacatecas, another famous Mexican silver camp. The younger of the two metamorphic series at Zacatecas has been shown by Burckhardt to be marine Upper Triassic, but the age of the older series, or the sericitic schists, at Zacatecas is unknown. The oldest rocks exposed in the silver camp of Guanajuato bear some resemblance to the younger metamorphic series of Catorce.

Unmetamorphosed Sedimentary Rocks

The sedimentary rocks overlie the metamorphic rocks with strong angular unconformity. From base upwards a generalized section of the sedimentary rocks follows:

1. Conglomerate, purple, green or brown, coarser below and finer above; the water-worn pebbles are chiefly quartz, of the underlying metamorphic series, and crystallines. The matrix is mostly quartz sand, firmly cemented. The conglomerate forms a prominent bench in the cañon walls. Its thickness is at least 100 ft. (30.5 m.).

2. Sandstone, argillaceous, below and shales, arenaceous, above; irregularly colored green, purple, brown or yellow. Strata exhibit a tendency to finely spaced jointing, which might lead to false supposition of true bedding-planes. Its thickness is about 400 ft. (122 m.).

3. Shales, marly, below and calcareous marls with thin interbeds of limestone above; grades upward into limestone; color gray, weathering pinkish or yellowish. Its thickness is perhaps 100 ft. (30.5 m.).

4. Limestone, heavy bedded, dark gray or dove-colored, fine-grained (as are all the limestones); has nodules and lenses of dark chert. Fossils are probably of lower Upper Jurassic (Oxford).[4] All the more productive ore deposits appear to be in this limestone. The limestone forms a prominent cliff in the north wall of the cañon near its mouth, crosses the cañon at its mouth, forms the summits rimming in the basin of the main town of Catorce, forms the top of the cañon walls below Catorce and is the lowermost limestone exposed in the cirque-like valley head above Potrero. Its thickness is at least 900 to 1000 ft. (274 to 305 m.).

5. Marls and thin limestones interbedded, with nodules of phosphatic(?) limestone; some of the limestone is rather coarse-grained; dark

[4] Preliminary studies of the fossils have been made by Dr. Carl Burckhardt and Dr. Emil Boese; the latter accompanied the writer during the four days spent in the field.

blue-gray, weathering to pinkish or ashy. Fossils of brachiopods (rhynchonelloids), pelecypods, and ammonites. Its thickness is about 100 ft. (30.5 m.).

6. Limestones, dark gray, medium- to thick-bedded, with nodules and lenses of black chert; ammonite fossils. Its thickness is at least 600 ft. (183 m).

7. Limestone, thin-bedded, with layers of marl, blue-gray, and nodular chert; the upper beds are limestone with thin layers of black chert and a relatively larger proportion of whitish marls; ammonites of upper Lower Jurassic (Portland). Its thickness is about 700 ft. (213.4 m.).

Members 4 to 7, inclusive, apparently give a complete section of all the Upper Jurassic. Middle Upper Jurassic (Kimmeridge) ammonites are found in the collections made. These members mainly outcrop in the drainage basin of the arroyo that runs north from Potrero.

8. Limestone, medium- to heavy-bedded, dark blue-gray, with chert nodules; ammonites of the Valangian division of the Neocomian (Lower Cretaceous). The upper limestones are thinner bedded, with thin layers of marls and fine sands, weathering pinkish, and containing Hauterevian (Lower Cretaceous) ammonites. It outcrops on east side of valley of Potrero where strata are sharply upturned and shattered, in lower walls, near mouth, of tributary gorge at least 1 mi. below the Refugio mine. Basal beds of this member may yield lowermost Cretaceous fossils. There may be higher Cretaceous strata to the east, but they do not appear to outcrop within the area of the mining district.

Structure

In general, the upper sedimentary series is folded into a rather broad dome. The strata dip westwardly from the upper town of Catorce and disappear under alluvium at the western foot of the range. The dip generally is eastward from the divide between upper Catorce and Potrero as far as the eastern divide of the valley running north from Potrero. There are a number of minor flexures in the broad domical structure and also quite a number of faults of only medium amounts of displacement. A number of these faults are easily seen in the bluffs around the upper town of Catorce, where the highly colored members of the sedimentary series are faulted against the limestone. All faults noted are of the gravity (normal) type. Most of the upper town of Catorce lies on a sunken block at or near the summit of the dome, the sides of the sunken block being formed by faults.

Igneous Rocks

1. Basalt or basic andesite forms a plug about 1 mi. northeast of the upper town of Catorce; it is found on the top of the ridge. The intrusive cuts the highly colored shales and sandstone member on the

southwest and the limestone on its other sides. The limestone strata are locally turned up at the contact and the color of the limestone has been changed to pink. One specimen of the igneous rock contained fragments of the arenaceous shale. The intrusive is very dark gray, almost black, and varies in texture from dense, aphanitic, and compact to amygdaloidal and agglomeratic. A smaller mass of what appears to be a similar rock was noted just west of the Potrero-Cedral road at the lowest crossing of the arroyo running north from Potrero.

2. Porphyry dikes, probably of quartz-monzonite or a related rock, cut the sedimentary series in a general north-south direction. The dikes vary in width from at least 1000 ft. (304.8 m.) almost to the size of stringers. They may possibly be offshoots of one main dike, toward which they are said to dip from both sides. They have phenocrysts of feldspar, often twinned, of varying sizes up to 2 in. (5 cm.) across. There are smaller phenocrysts of quartz up to ½ in. (1.27 cm.) across. The feldspar phenocrysts have sharp crystal faces and outlines; the quartz ones do not. The ground mass is finely crystalline. The dike rock is always altered to a greenish, yellowish-green or brownish-green, and the feldspar is extensively kaolinized. The porphyry is less resistant to erosion than the sedimentary limestones; hence the outcrop of a dike can be traced by a depression of the surface, generally either an arroyo or a shallow trench. There was not the slightest trace of contact metamorphism visible to the unaided eye at contacts of the dike rock with any of the sedimentaries.

Spurr and Garrey have described two bodies of quartz-monzonite-porphyry near a large fault with at least 1500 ft. (457 m.) displacement at the eastern base of the Cerro del Fraile, about 9 mi. (14.5 km.) east of Potrero. At this place is situated the Dolores mine, a contact-metamorphic deposit of copper minerals. There is a possibility that a deep-seated intrusive igneous mass underlies the Catorce district and that the dikes may be apophyses of this mass.

Mineral Deposits

The primary gangue minerals are calcite, quartz, rhodochrosite, and possibly some others, including anhydrite. The primary ore minerals are mainly sulfides of iron, iron and copper, lead, zinc, arsenic, antimony and bismuth. Mercury sulfides are found south of the main district. Subsequent alteration of the sulfides formed, principally in the oxidized and leached zones, the haloid silver minerals, the double salts of silver, and possibly even more complex silver minerals.

The main deposits are fissure veins in limestone No. 4 of section; the intersection of veins were the sites of rich bonanza oreshoots. The fissure veins are probably not all along fault lines. Some of them may

follow joint planes. There are also quite a number of pocket deposits, as in the other regions of limestones in northeastern Mexico. Most of these pockets occupy the sites of former solution channels and cavities. No deposits appear to have been found in the shale in which, from the incompetent nature of the rock, there probably could not have existed open fissures or channels. However, mineralizing gases may have penetrated these arenaceous shales and encountered more favorable conditions (physical, chemical, or both) in the overlying limestones. According to R. deW. Armit, general manager of the Dolores and Refugio mines, the dikes are later than the veins, the latter being cut by the former. Judging from the physiographic history of the region, the primary ores were originally deposited probably about 10,000 ft. (3048 m.) beneath the surface.

Date of Mineralization

No data bearing directly on this problem can be procured within the limits of the Catorce district. There appear to be two probable epochs of mountain-building in this part of Mexico, the one at approximately the end of the Cretaceous (Laramide) period and the other about the middle of the Tertiary. The evidence for the first epoch is a reported folding and subsequent erosion of the Cretaceous before the deposition of the Eocene along the foot of the eastern flanks of the Mexican Cordillera, not more than 100 mi. (160.9 km.) east of Catorce. The evidence for the mid-Tertiary mountain folding is based on three things: (1) mountain folding of known mid-Tertiary date in New Mexico; (2) a similar state of physiographic development in the same kinds of rocks and in substantially the same climate in both northeastern Mexico and in New Mexico; (3) the occurrence of pebbles of rhyolite in probable later Miocene marine strata of the Gulf coastal plain of the State of Vera Cruz, the eruption of the rhyolite being assumed to have been contemporaneous with or only slightly later than the mountain folding. The physiographic evidence is perhaps the best, for the present mountain ranges of sedimentary rocks in northeastern Mexico are anticlines. To be sure, the rocks of the mountains are limestones, most resistant in the arid climate, but the fact that the mountains are still the anticlines and the intermontane depressions the synclines call for a stage of topographic development that it hardly seems possible could have taken more than half the entire Cenozoic time to bring about.

The basalt or basic andesite may conceivably have been intruded at any one of several times between the Cretaceous and the present, for those rocks were erupted in the Mexican region at different dates during the Cenozoic era. But the dike rocks belong to a petrographic province of rocks of like composition that are found all the way from the San Juan Mountains of southwestern Colorado to central Mexico, the intrusion

of which appears to have been substantially contemporaneous with mid-Tertiary deformation, and which are viewed by practically all their students, although not on absolutely conclusive evidence, to be about middle Tertiary in age. However, we do not know that the Catorce ore deposits are genetically related with either of the visible intrusive rocks.

Possible Localization of Limestone Ore Deposits of Northeastern Mexico in Anticlinal Structures

All the workable ore deposits noted by the writer in the limestone rocks of northeastern Mexico (in the region east of the great territory of extrusive and intrusive rocks of the misnamed Mesa Central) are in anticlinal structures. However, it must be said that ore deposits may occur in the synclines where there is little hope of discovering them; for the synclines occupy the low lying areas where the bed-rocks are covered with alluvial debris. The ore deposits noted are generally along the axes of the major anticlinal structures and subordinately in zones of minor crumplings within larger anticlinal structures. These are, first of all, places where there are open spaces in the rocks. In this respect they are possibly somewhat analogous to the saddle reefs of Bendigo. Second, the limestones were formerly overlain by several thousands of feet of upper Cretaceous rocks, which were largely shales. It is possible that ascending mineralizing solutions of gases may have penetrated the pervious limestones, reached the highest places in the structures, and there deposited the minerals under the overlying impervious shales. It is likely that there were more open spaces in the anticlines than in the synclines. Third, the chemical composition of the limestones may have been more suitable for deposition of minerals than sedimentary rocks of different composition. Fourth, the anticlinal axes were places of least resistance, along which the passage of igneous intrusions was facilitated.

The Dip Needle in Stratigraphy*

By H. R. Aldrich,† Madison, Wis.

(Lake Superior Meeting, August, 1920)

This paper presents some of the results obtained during the field season of 1919 while mapping, in detail, the stratigraphy of the Gogebic Range in Wisconsin. The detailed stratigraphic section for the range was first published, in 1919, by W. O. Hotchkiss,[1] State Geologist of Wisconsin. The idea of the continuity of the definite succession thus established was the result of underground work in the productive part of the range in Michigan and Wisconsin, where most of the required data also were obtained. The bulk of the range in Wisconsin is heavily covered by drift; in fact, there is but one satisfactorily exposed section across the entire formation, few partial sections, and only scanty outcrops of any size. Evidently mapping of the detailed members of the formation could only be established at a few points; and if from these small areas the unexposed portions were to be mapped by projection, the only apparent method for projection was straight-line interpolation modified or supported by data from the scanty outcrops and from test workings.

It was found that by tracing a magnetic line, the position of which in the formation may be determined from test pits and outcrops, the projection of that member through the drift-covered areas can be accomplished. The magnetic line serves as a datum horizon within the formation to which reference may be made in correlating outcrops and ledge matter thrown from test workings. The foot wall is easily located also, and from the two datum horizons thus available and the known proportional thickness of the several members in the type sections, outcrops and artificially exposed ledges can be correlated by lithological characteristics. As a result, the tracing of magnetic lines became the basis of our work on the stratigraphy and we were able to carry the mapping of detailed members throughout the length of the range. It was found that the magnetic horizon at the base of the Tyler slate was most easily followed by the dip needle; that discontinuity caused by cross-faulting could be established; that folding along the strike could be detected in a repetition

* Published by permission of the Director, Wisconsin Geological Survey

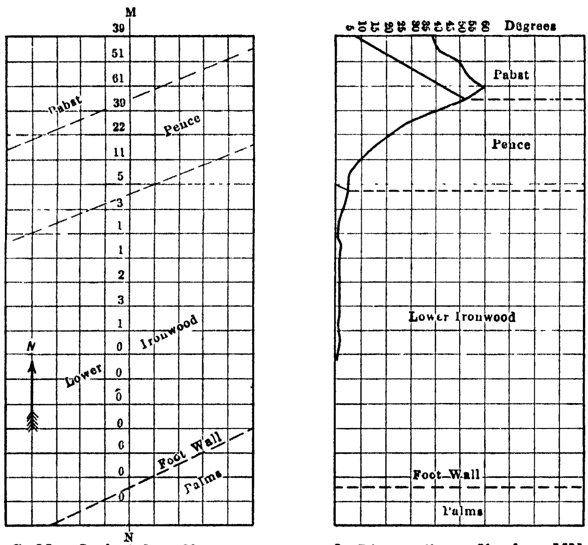
† Geologist, Wisconsin Geological Survey.

[1] W. O. Hotchkiss: Geology of the Gogebic Range and its Relation to Recent Mining Developments. *Eng. & Min. Jnl.* (1919) **108**, 443.

of this horizon; that cross-folding could be traced; and that zones of oxidation in connection with faults, a very important condition in relation to ore formation, could be detected. It was also clearly evident that there were limitations to the use of the dip needle.

The Magnetic Datum Horizon

The magnetic datum horizon established for control in mapping lies in the Pabst member at the base of the Tyler slate. To illustrate the magnetic relief of this horizon, a page from a field notebook has been



a Map-Scale-1 div.= 20 paces *b* Dip needle profile along MN

Fig. 1.—Pages of field notebook, showing dip-needle readings and curves.

reproduced in Fig. 1*a*. The figures along the north-south line are readings on the dip needle taken upon a north-south traverse from a point in the Tyler to the foot wall. The Pabst beds were identified in an exploration trench. In Fig. 1*b*, a curve of the dip-needle readings has been drawn to emphasize the sharp contrast between the attraction over the Pabst member, and that over the rest of the formation. The needle used normally reads 12° in areas of no attraction and on the curve the vertical ordinates give the variation from the normal reading. The ease with which a member giving rise to attraction of this type may be followed is evident. In the westward extension of the range, conditions affecting the contrast in attraction between that in the Pabst and the rest of the

formation are somewhat different and difficulties were encountered; weakening of the attraction at the location of cross faults increased the difficulties. However, for most of the distance from east to west the horizon was readily traceable.

Cross Faults

The existence of cross faults on the Gogebic Range is well known and their influence on the concentration of ores has been recognized. The detection of the faults when drift precludes their determination by other methods rests on the employment of magnetic instruments. That

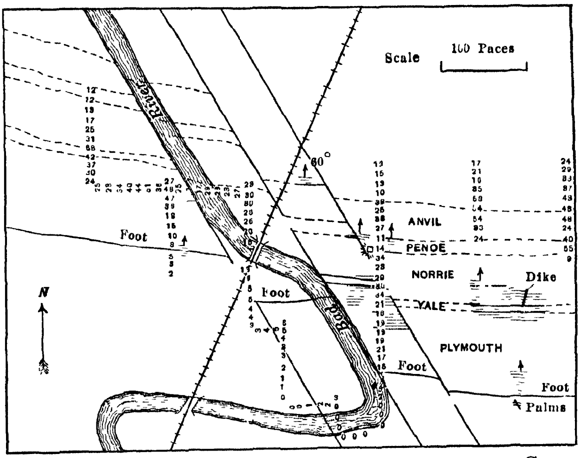


Fig. 2.—Determination of faulting at Penokee Gap.

this can be done is shown in Figs. 2, 3, and 4. To test the practicability of the method, work was done on an area where faulting was proved by mine workings and where test pits were sufficiently numerous and conveniently located to aid in interpretation.

Fig. 2 shows the solution of the faulting at Penokee Gap by magnetic methods. Two and probably three blocks are present. Between the outer two, the area is in the valley of the Bad River where outcrops are missing and the mantle is heavy.

Fig. 3, illustrating the determination of three faults by means of magnetic attraction, is explained fully to show the methods of arriving at the conclusions. The structure of this part of the Gogebic Range is simple. The dip is about 60° to the north, there is no folding whatever, and the strike of any one member is so regular that, on the scale to which this figure is drawn, its trace in the rock surface beneath the drift may be projected with a straight-edge. The surface trace and the magnetic line

of the member followed by means of its magnetic attraction should be identical. If the formation is dislocated by a cross fault—and cross faults are known to be frequent on the range wherever mine workings make their direct observation possible—the line of magnetic attraction, which is the surface trace of the given member, will show this offset.

In Fig. 3, the outer line marking the division between the Tyler and the Anvil is the line of magnetic attraction followed. Its regularity of strike and the ease with which it was traced by means of the dip needle require no comment. At three points within the limits of this area there was a repeated offset to the north, indicating as many cross faults. There was no departure from this method whether the magnetic horizon followed lay in the hanging wall or along the foot wall, as in Figs. 2 and 4.

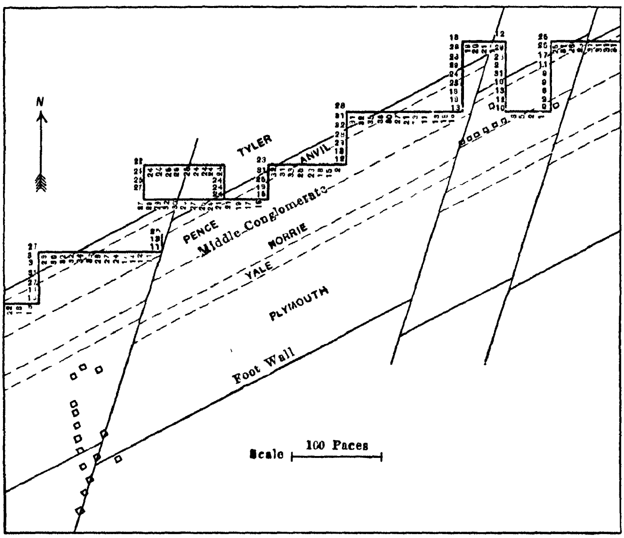


FIG. 3.—DETERMINATION OF CROSS FAULTS BY MAGNETIC ATTRACTION.

To determine the strike of the fault plane, additional data were required which, in the case of the area of Fig. 3, consisted of test-pit data as shown, a knowledge of the strike of faults in properties not far removed, and indications of such faulting contained in the level maps of the mine formerly worked on this property.

In the case of the middle fault, the dislocation of the line of magnetic attraction indicated a fault. Faults in properties not far removed strike roughly parallel to the eastward pitching dikes, that is northeastward. The fault plane does not cut through the line of test pits on the conglomerate and passes between the two north-south magnetic traverse just above the line of test pits. Therefore its location is established between

limits and is given the northeast strike as suggested by the strike of faults in nearby properties. One additional indication of this location is the weakening of the magnetic attraction along the east-west traverse through which the fault is drawn. This weakening is possibly due to oxidation of the magnetic components of this horizon by waters working along the fault.

The fault on the west side of the figure was detected in the dislocation of the magnetic line. From examination of material thrown from the test

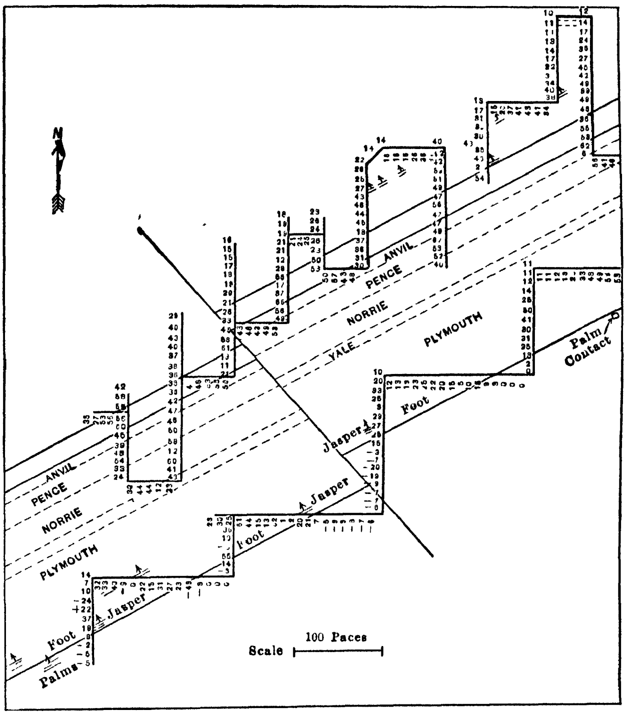


FIG. 4.—DETERMINATION OF CROSS FAULT, REPETITION OF MAGNETIC HORIZON BY FOLDING.

pits in the foot-wall area, the probable strike of the fault was established and found to agree with that of the middle fault.

The third fault was detected in the dislocation of the line of attraction, and since there were no further data in support, its strike was assumed to be parallel to the others.

The location of the foot wall in Fig. 3 is established by test-pit data in connection with the fault on the west side. In view of the regularity of the formation here, we were justified in measuring the width of the forma-

tion between the magnetic horizon and the foot wall and projecting this across the faulted area. The objection to this arises from the observation that a fault in the hanging wall is many times represented by a mere roll in the foot wall. The displacement of the foot wall in Fig. 4 was established by following this horizon by means of the magnetic attraction, and this determination was supported by outcrops.

Fig. 4 is a magnetic map of a portion of the range several miles west of that shown in Fig. 3. Mines are absent here, and there are no means for checking the determination from mine drawings. At this point it was far more convenient to follow the foot wall magnetically than the original horizon in the hanging. The fault is well shown by the magnetic attraction. The foot wall is projected westward to the fault by magnetic attraction supported by outcrop data east of the area presented and is picked up west by magnetic attraction supported by outcrops.

Longitudinal Folding in the Hanging Wall

It has been stated that toward the west difficulties were encountered in following the Pabst horizon because of a lack of magnetic relief arising from more uniform distribution of magnetite throughout the Ironwood. Also, in the description of Fig. 4, it was stated that the foot wall was more readily followed for purposes of detecting faults than was the Pabst line. The magnetic conditions found in the hanging wall in the area of Fig. 4 illustrate the present contention that repetition of the Pabst by longitudinal folding can be detected by the dip needle. Conversely, the repetition of the horizon, as shown by magnetic attraction, can be explained as due to folding down the dip. This difficulty in interpretation of magnetic attraction was not explained definitely until actual exposures were found which showed the folds.

Cross Folding

In the west end of the range, deformation is greater than in the east. The turning of the magnetic line confirmed by topographic data and assisted by information from test pits permitted the mapping of cross folds west of Tyler's Fork.

Zones of Oxidation Along Fault Zones

The value of information concerning the oxidation of the formation along cross faults cannot be denied in the light of the influence faults have in concentrating the iron ores. In the following of the magnetic line, it was disclosed that when the attraction showed signs of considerable weakening a cross fault might soon be expected. In fact, many of the more certain faults show pronounced weakening of the attraction especially on the east side. Fig. 2 illustrates this fact to a certain extent.

Limitations

The writer makes no claim for the dip needle as an instrument for final solution of all problems. As an instrument auxiliary to the methods commonly in use for mapping and determining structures in a drift country, its possible utility is far greater than has been commonly considered even in the iron-ore districts. A few of the more outstanding limitations may be cited. Attention has been called to the difficulties arising when the magnetic horizon has little or no magnetic relief. The difficulty here lies in the fact that several magnetic crests may occur at close intervals and the identity of any one can be established with certainty only when outcrops are found. This offers great difficulty, especially when cross faults are indicated because, due to oxidation or brecciation or other differential effects incidental to faulting, the identity of the magnetic datum may be lost in the local exchange of magnetic dominance among the several lines. In crossing the fault, it is difficult to make certain of picking up the proper line from the several that occur.

In a formation having a gentle dip, the following of a sharp line is practically impossible since the trace of the bed in the surface, even though it enjoys excellent magnetic relief, is not a sharp line but a belt sometimes of considerable width.

Extreme deformation by faulting and folding offers a handicap almost insuperable unless unlimited time can be spent and outcrops are available. This is especially true when the formation as a whole is uniformly magnetic.

Conclusion

The dip needle and its complementary instrument, the dial compass, offer a service to the geologist that has not yet been fully appreciated. This is due to two things. In the first place, some geologists have demanded that the magnetic instruments solve geological problems fully or not at all. In the second place, the fundamentals of the instruments and their use have not been understood. It is firmly believed that the use of the needle demonstrated here can be employed more widely and to good advantage by mining companies in the Lake Superior district, and that similar service can be rendered in other districts where the rocks are of different character.

DISCUSSION

H. R. Aldrich.—While magnetic instruments are not instruments of precision, by standardizing them frequently, with well known bases and checking by other means, the dip needle and dial compass are capable of far greater use than they have at present.

W. O. Hotchkiss,* Madison, Wis.—Mining engineers and geologists are neglecting a most valuable means of following up details of geology of economic significance in the various mining districts. No two rocks are physically exactly the same in magnetite content and if we have instruments that will detect that difference, we can follow the contact between even such weakly magnetic rocks as shale and limestone by that magnetic difference.

The difference in the magnetite content of rocks in many cases is sufficient for the rocks to be followed with a great degree of definiteness with the ordinary dip needle or dial compass. That particular property of rocks is the only one that at present can be detected through cover. Mr. Aldrich has done a fine piece of work in following, in considerable detail, in a covered area, the geology of the Gogebic Range in Wisconsin. This will save thousands of feet of drilling. But the method is applicable in other parts of the country as well. It is possible to trace almost any contact between an igneous rock and any other rock, as the difference in magnetite content is usually well marked in such cases.

Carl Zapffe,† Brainerd, Minn. (written discussion).—Any one contemplating a magnetic survey with a dip needle should study well the remarks under the heading "Limitations." The dip needle is most helpful, but if its limitations are not fully understood it should be laid aside or left for the more studious to use. In the long run magnetic readings are, at best, only a guide. In some fields, they indicate what to leave alone; in others, they point to the place suitable to make a beginning for other work. Complete results are impossible unless other positive data are available, such as are obtained from outcrops, test pits, or mine workings. Conditions influencing variations on dip needles are so numerous that it is quite necessary to have outcrops, test pits, or mine workings to tell what the many possible and peculiar combinations in the readings may indicate, much less mean, and enable predicting the details of concealed structures with some degree of reliability.

One has but to read carefully Mr. Aldrich's presentation to note how regularly his interpretation of magnetic readings is really the interpretation of observations made on nearby outcrops or in the mine workings; where the latter are lacking, his deductions lose rapidly in value because they lose in definiteness, regardless of the fact that the general structure in the Gogebic district is known to be far from complicated.

Emil A. Kronquist,‡ Duluth, Minn. (written discussion).—In his summary, Mr. Aldrich draws the following conclusions: (*a*) That discontinuity of magnetic lines caused by cross faulting can be established;

* State Geologist of Wisconsin.
† Geologist, Northern Pacific Ry. Co.
‡ Assistant Geologist, M. A. Hanna & Co.

(*b*) that folding along the strike can be detected in a repetition of this horizon; (*c*) that cross folding can be detected; (*d*) that zones of oxidation in connection with faults can be detected. The writer agrees with these conclusions, but believes that interpretations based on magnetic work must be controlled by the limitations of such work.

In the article referred to by Mr. Aldrich,[2] Mr. Hotchkiss proposed that the various horizons were rather constant and definite beds that could be traced for great distances from one end of the range to the other. Various geologists and engineers doubt whether the stratigraphy, as suggested by Mr. Hotchkiss, can be projected for any appreciable distance. Many have found it difficult to trace these horizons from point to point, even in the best known parts of the district. The writer believes that the horizons of iron formation are lens-like in character, as are all known sediments, and vary rapidly in thickness from point to point, and that

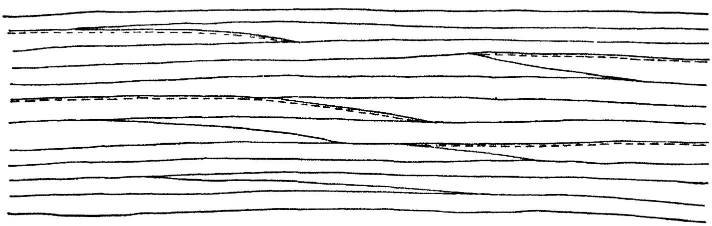


FIG. 5.—ARRANGEMENT OF LENSES AND THEIR CORRESPONDING MAGNETIC LINES.

definite beds do not maintain uniform characteristics except for relatively short distances. Fig. 5 shows the relation of magnetic lines to the boundaries of any given horizon under this condition of sedimentation. This consideration must be taken into account before assuming that Mr. Aldrich has been able to extend stratigraphic horizons from one end of the range to the other.

To draw conclusions with the degree of certainty developed in the paper, it is necessary to take into account certain assumptions and certain limitations. In order to tie the crests of magnetic maxima into structural units one must assume that a given horizon is uniformly magnetic throughout, and that it is the same horizon and not one 10, 50, or 100 ft. stratigraphically above or below, which may locally be strongly magnetic. As a matter of fact, a given horizon usually proves magnetic for relatively short distances for any of the following reasons:

1. A given horizon does not continue indefinitely because its original deposition was controlled by the prevailing conditions of sedimentation of that particular area. Its persistence is controlled by changes in the

[2] *Eng. & Min. Jnl.* (1919) **108**, 443.

shore line and the supply of sedimentary materials, and the shifting of lagoons, channels, etc. in shallow waters.

2. The erratic distribution of the iron content and the carbonaceous material or other reducing agents in the original deposition of the iron formation would affect the subsequent nature of the formation. If it is assumed that metamorphism was equally effective throughout the horizon (which is never the case), portions of it would be more highly magnetic locally than others, depending on the original content and distribution of the magnetite-forming constituents.

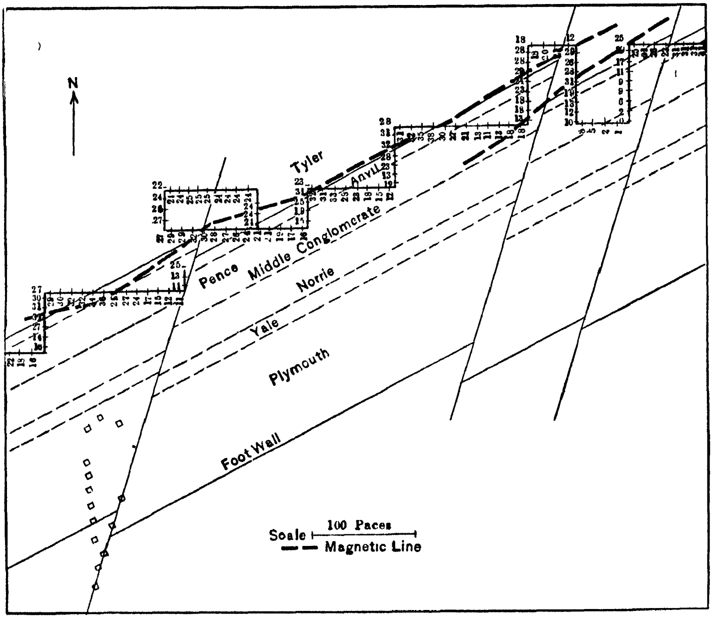


FIG. 6.—AFTER MR. ALDRICH'S FIG. 3.

3. Any horizons subjected to intense folding or intrusion are locally more highly metamorphosed than others. If it is granted that the magnetite-forming elements were originally equally distributed throughout a given horizon, given localities are bound to be more highly metamorphosed than others, according to their position on the structural units.

4. Although originally given horizons may have been uniformly magnetic for considerable distances along the strike, subsequent erosion and oxidation along fracture and fault planes may have rendered portions of the horizon relatively non-magnetic in contrast with other portions of the same horizon. Deep pre-glacial erosional valleys now filled with glacial drift would cause variations in the dip needle adreings, and if not

properly taken into account, would cause complication in connecting the various magnetic maxima.

These limitations are generally recognized and have been encountered while doing magnetic work in all the Lake Superior iron districts, including the Wisconsin district under discussion. They must be taken into account before conclusions as to the geologic structure of any given area can be properly interpreted.

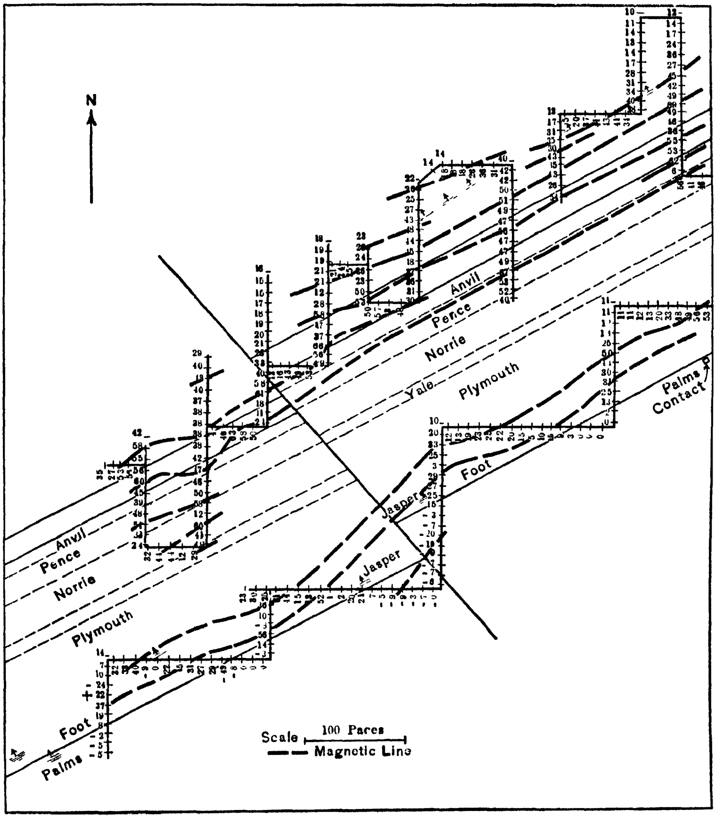


FIG. 7.—AFTER MR. ALDRICH'S FIG. 4.

A careful study of Figs. 3 and 4 of Mr. Aldrich's paper leads to the belief that these maps are either incomplete or that the structures have been worked up from other than the magnetic data. The conclusions are sound and logical, but the facts presented seem forced to meet the conclusions. For example, in Fig. 4, west of the fault plane, the contact between the Pabst and the Anvil is marked by a strong magnetic crest,

with another crest at the base of the Pence. Immediately east of the fault, the upper magnetic crest has moved to the top of the Pabst and the lower magnetic crest has moved to the top of the Pence. In the extreme eastern end of the map, the upper crest is well out in the Tyler slate and the lower crest is well into the Anvil. These maximum magnetic attractions are not in alignment with one another, and if one were to follow the conclusions as listed in Mr. Aldrich's paper, he could have several faults cutting this area instead of one.

In forming conclusions from magnetic data, it is always advisable to arrive at these conclusions by passing from the simple possibilities to the complex. Fig. 4 redrawn to show the possible simple connections of the magnetic maxima is shown in Fig. 7. With only the magnetic information at hand, it is easy to join the magnetic maxima in an entirely

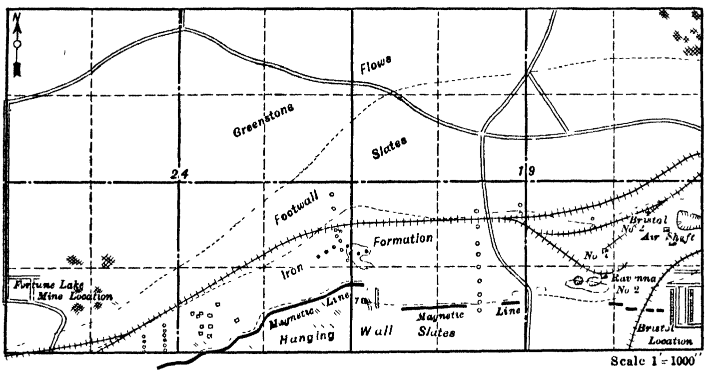


FIG. 8.—MAGNETIC DISPLACEMENT NEAR FORTUNE LAKE MINE, CRYSTAL FALLS DISTRICT

different fashion from Mr. Aldrich's, showing a simple and normal structure, with no faults. A study will show that the magnetic lines of the foot wall are continuous and parallel, whereas the magnetic lines in the upper portion of the formation are not continuous for even the short distance represented in the sketch. While doing detailed magnetic work on the Gogebic Range, the writer has many times found that the magnetic conditions were so complex that readings taken two feet apart would show extreme variations, variations which had no apparent relationship.

The writer agrees that zones of oxidation can be detected along faulted and fractured areas, but he has observed many localities where this is not true. Figs. 3 and 4 are good examples of the latter. The westernmost fault plane in Fig. 3 (on the hanging) passes through an area of the highest magnetic attraction, as does also the easternmost fault; the central fault shows little if any loss of magnetic property.

Careful magnetic surveys are useful in following folded horizons, often outlining structures not appreciated from a study of limited outcrops. The Crystal Falls district offers many opportunities for a careful study of this type, often developing suggestions of structural conditions that are extremely important from an exploratory standpoint. The Crystal Falls district is a region of intense folding, with a normal amount of fracturing, and only minor slippage, with no faults of appreciable magnitude. One example of an apparent displacement of a magnetic horizon may be cited as follows: A well defined magnetic line representing the contact of the hanging-wall slates with the iron formation at the Fortune Lake mine property can be traced slightly north or east from the SW. corner of Sec. 24, 43-33 to a point a little east of the western

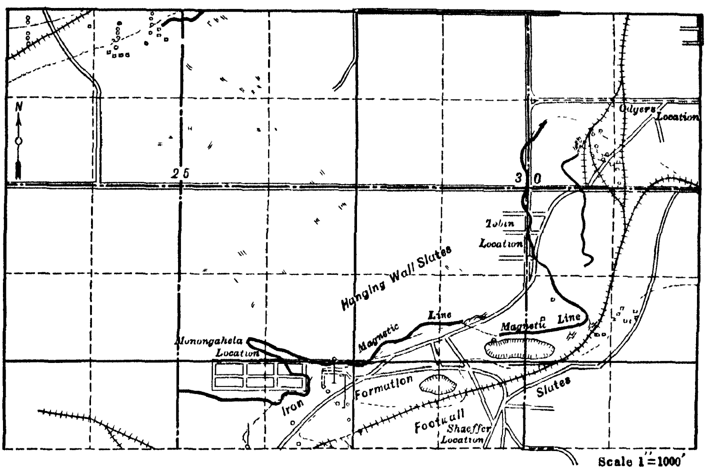


FIG. 9.—MAGNETIC DISPLACEMENTS NEAR TOBIN MINE, CRYSTAL FALLS DISTRICT.

property line of the Ravenna mine, where this magnetic line ends abruptly. Continuing to the east, but with an apparent offset of approximately 300 ft. to the south, this magnetic horizon can be traced intermittently for nearly a mile. Recent exploratory work has proved the existence of a large drag fold in this locality. The severe fracturing and the synclinal structure proved readily accessible to the movement of ground waters, causing oxidation. It is important to note that the contact horizon is non-magnetic in this particular locality.

Another example of a displacement in a magnetic horizon will be found just north of the Tobin mine. A strong magnetic line can be traced eastward for ½ mi. from an outcrop of magnetic black slate at the old Monongahela mine to a point slightly northwest of the Tobin mine, where this line ends abruptly. A magnetic horizon is located 300 ft.

(91.4 m.) south and 500 ft. east, which in turn can be traced continuously for more than ½ mi. to the north and east, passing over an outcrop of material very similar to that found at the Monongahela mine. A study of the structural conditions in the western portion of the Tobin mine, however, indicates a large gentle synclinal structure associated with more or less fracturing, with no evidence whatever of appreciable movement by faulting, indicating that the magnetic horizon lost its magnetic properties on the crest of the anticline to the west of the Tobin syncline and resumed its magnetic properties on the crest of the anticline to the east of that structure—which, through oxidation and leaching by the movement of ground waters down the syncline, rendered the hanging-wall slate non-magnetic.

A further example of apparent displacement in magnetic lines will be found in Section 20, 43-32, immediately south of the power plant. Three parallel magnetic lines can be traced from the western section line to a point approximately 100 ft. west of the Paint River, where these lines end abruptly; 700 ft. to the north another set of three parallel lines were encountered, the western extremities of which are sharp and abrupt. It is a peculiar coincidence that in these two sets of three parallel lines both sets are nearly equally spaced between the magnetic horizons. Without detailed information from the adjoining mining properties, one coming directly from the Gogebic Range would undoubtedly place a fault plane through the area where these lines end so abruptly. But careful correlation shows that the last-mentioned lines are stratigraphically below the first-mentioned set, and that the sharp break in the magnetic lines is due to severe fracturing and subsequent oxidation and not due to faulting.

H. R. Aldrich (author's reply to discussion).—Mr. Kronquist says that the writer's maps "are either incomplete or the structures have been worked up from other than the magnetic data," and that "the facts presented seem forced to meet the conclusions."

These points are well taken, for the maps are lacking in many of the features that, combined with the magnetic data, have been used to establish structure. For example, topography has been omitted and topography was of great assistance in determining structure in this work. Furthermore, outcrops beyond the limits of the maps gave the clue to the interpretation of the magnetic attraction in Fig. 4. In the same figure the cross fault is shown up by the dislocation of the jasper of the foot wall. Two outcrops are shown, one on either side of the fault, and with these in mind there is no reason for redrawing the magnetic lines as was done. The magnetic data have been used "for all they are worth" but none of the structures have been established on such data alone.

In regard to Fig. 6, these faults are shown by test pits and confirmed

by mine blueprints. Before hoping to detect cross faults by magnetics, considerable experimentation was done on the surface of a property where faulting was shown by workings and test pits. In working westward it became more and more obvious that early prospectors had followed the same line of control we were following (magnetic tracing of the Anvil-Pabst contact) and that discontinuities were interpreted as offsets of the horizon by faulting. Test pits cluster around several such localities and extend from foot to hanging wall in several instances. In more than one instance, the test pitting still shows the cross faulting.

It is the writer's belief, as a result of the field work and the examination of hundreds of specimens megascopically and microscopically, that the five subdivisions extend throughout the Wisconsin end of the Range and that they are substantially of uniform characteristics. The writer cannot agree with Mr. Kronquist's statements that the Ironwood is of lenticular structure in any sense as represented by Fig. 5. Nor that all known sediments are similarly lenslike, vary rapidly in thickness from point to point, and do not maintain uniform characteristics except for relatively short distances. The greater number of differences between Mr. Kronquist's ideas and those of the writer are inherent in the conception of the original sedimentation of the Ironwood.

Rate of Formation of Copper-sulfate Stalactites

By Graham John Mitchell, Warren, Ariz.

(New York Meeting, February, 1921)

In May, 1919, a crosscut on the 1400-ft. level of the Briggs mine, a Calumet, & Arizona property at Bisbee, Ariz., penetrated a deposit of pyrite and chalcopyrite that had replaced quartzite and limestone and was cut, especially in the quartzite, by numerous fractures. As the drift advanced through these fractured portions, they were found to be free from copper-sulfate minerals although "sooty" chalcocite was present in the fractures and also as coatings and replacements of pyrite. However, sulfate soon made its appearance in certain fractures, filled them, and began to form stalactites. Several centers of incipient growth were noted for subsequent study, observations in the drift at frequent intervals during the past 17 months affording the opportunity of determining the rate of formation of the stalactites.

The one to attain the greatest growth during the period of observation measured 27 in. long and averaged 1 in. in diameter. A close study showed it to be composed of small interlocked crystals of chalcanthite radiating from a small central tube-like opening. The triclinic form of the crystals was evident with prism and pinacoid faces recognizable.

Assuming that the supply of copper-sulfate solution was constant during the 17 months—and such an assumption is supported by the fact that the number of drops of excess solution falling, per minute, from the tip of the stalactite at the times of observation checked closely—the stalactite, it has been estimated, grew at the rate of 1.58+ in. per month.

DISCUSSION

James F. Kemp,* New York City.—J. S. Curtiss, in the monograph on Eureka, Nev., says that a stalactite formed in limestone grew ⅝ in. in three weeks and continued to grow as long as it had a drop of water or solution at the lower end. Another grew at the rate of ⅜ in. in the same time. In the case cited by Mr. Mitchell, the growth was about 1⅛ in. in three weeks.

A year ago we had an instance of the formation of cubes of galena on a railroad spike. A mine in northeastern Oklahoma had been idle about two years but on account of pumping of neighboring properties, the circulation of water was active throughout the drift. A cube of galena,

* Professor of Geology, Columbia University.

about ½ in. on the edge, had formed on a railroad spike lying at the bottom of the drift. That led to quite a discussion on the rapidity of the growth of galena in the southeastern Missouri lead and zinc district.

Occasionally, we observe some peculiar and interesting things in the gash vein districts in Wisconsin and the neighboring parts of Illinois. I have one or two stalactites, with some pyrites, that are a little less than ⅛ in. in diameter and 2 or 3 in. long, with a hole through the center, but they turn at right angles for about ½ in. at the lower end and grow in exact right angles to their original course. It is quite difficult to understand how the current of the water could be turned at right angles.

Our general feeling about the precipitation of ore minerals, as we have studied geology, has been to shorten the time that we thought was required; and, possibly, the formation of these soluble and oxidized compounds in newly opened ground will tend somewhat in that direction. Naturally, mine drifts drawing water at a high rate from the surrounding and overlying rocks afford a much more rapid circulation than we can suppose to have occurred during the original deposition of the ores.

Judging the Quality of Portland Cement

By R. J. Colony,* New York City

(New York Meeting, February, 1921)

The failure, or disintegration, of concrete in structures, even when the cement, sand, and coarse aggregate used have passed satisfactorily all tests and inspections, is not uncommon. Such failures occur even when proper methods of mixing and placing have been used and where weather conditions were correct. In these cases, the origin of the trouble must be sought in the cement; therefore some additional mode of determining the quality of cement must be employed.

The problem is to find a simple, usable, easily applied method that will supplement the standard tests. The solution of this problem involves a knowledge of the chemical composition of Portland cements, also of their constitution (or componental composition), as the behavior of a cement, when gaged, is dependent on the physical and chemical characters of the compounds of which it is composed, the proportions existing between them, and the fineness of grinding. It is likewise essential to know the character, behavior, and composition of the hydration products of Portland cement.

The most promising method of attack makes use of chemical, mechanical, and petrographic methods. Chemical, by determining quantitatively not only the usual constituents, but the carbon dioxide and water. Mechanical, by determining the fineness with the air analyzer,[1] which separates into fractions of various grain dimensions all of that portion which in the ordinary sieve test passes the 200-mesh sieve and constitutes at least 75 per cent. of the total cement. These fractions may be further studied chemically and petrographically and micrometric measurements can be made of the grains. Petrographic, by the method of immersion, and by making thin sections from test pats of both neat cements and standard mortars.

Normal Portland cement is composed of mechanical mixtures of definite chemical compounds having constant chemical and physical properties. In the order of their cementing qualities these are: Tri-

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* Instructor in Geology, Columbia University.

[1] J. C. Pearson and W. H. Sligh: An Air Analyzer for Determining the Fineness of Cement. *Tech. Paper* 48, U. S. Bureau of Standards (1915).

calcium silicate, $3CaO.SiO_2$, tricalcium aluminate, $3CaO.Al_2O_3$, tricalcium ferrite, $3CaO.Fe_2O_3$, and calcium orthosilicate, beta form, $2CaO.SiO_2$. In addition, there is a certain quantity (generally about 3 per cent.) of gypsum. The magnesia, MgO, is in a state of solid solution in the other components and is apparently without much effect in the quantities found in cements. Both potash and soda exist to a small degree, and flakes of quartz from the ball mills, metallic iron from the machinery, particles of semi-fused coal ash from the fuel, etc., are sometimes found.

A long series of studies on the ternary system,[2] lime-silica-alumina, and the system alumina-silica-magnesia, as well as studies of binary systems of the same components, by physicists, physical-chemists and chemists of the Geophysical Laboratory, the Bureau of Standards, and different universities, and a comparison study of Portland cements have shown quite definitely that the three constituents that form the bulk of normal Protland cement—lime, CaO, silica, SiO_2, and alumina, Al_2O_3—exist as fixed components with definite chemical composition and constant optical properties.

A study of the analyses of standard brands of normal Portland cement will show that their percentages of lime, silica, and alumina amount to over 90 per cent. Perfectly sound Portland cement has been made of these three constituents alone; hence if the percentages of these constituents are recalculated to a basis of 100 per cent., neglecting the ferric and other oxides that are always present in small amounts, the resulting percentages may be plotted on the ternary concentration diagram.

Points thus plotted will fall within a restricted triangular area within the diagram, with three compounds forming the apices of the small triangle; viz., tricalcium silicate, $3CaO.SiO_2$, tricalcium aluminate, $3CaO.Al_2O_3$, and dicalcium silicate (or calcium orthosilicate), $2CaO.SiO_2$, beta modification. Differences in the percentages of lime, silica, and alumina, within certain limits, do not cause the appearance of a fourth and new compound; they merely change the proportion between the three compounds mentioned; this is an important point. If, however, the difference in percentage composition is great enough, on recalculating the percentage of the three constituents (SiO_2, Al_2O_3, CaO) to a 100-per-cent. basis and plotting the result, the point will fall outside of the area mentioned and different compounds may be expected to occur, some of which may be non-hydraulic.

The results of about fifty analyses made in the laboratory of the Board of Water Supply of the City of New York have been recalculated and plotted on the triangular concentration diagram in the manner described (some of the dots cover a number of plotted points); all the points so

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[2] G. A. Rankin and Fred. E. Wright: The Ternary System $CaO\text{-}Al_2O_3\text{-}SiO_2$. *Am. Jnl. Sci.* [4] (1915) **39**, 229.

plotted fall within the triangular area mentioned except two, *A* and *B*, Fig. 1. One of these *A* is a cement with abnormally high silica (33.01 per cent.) and low lime (52.73 per cent.); the other *B* falls within the limits of the composition usually set for Portland cement, but it also is a little too high in silica and too low in lime.

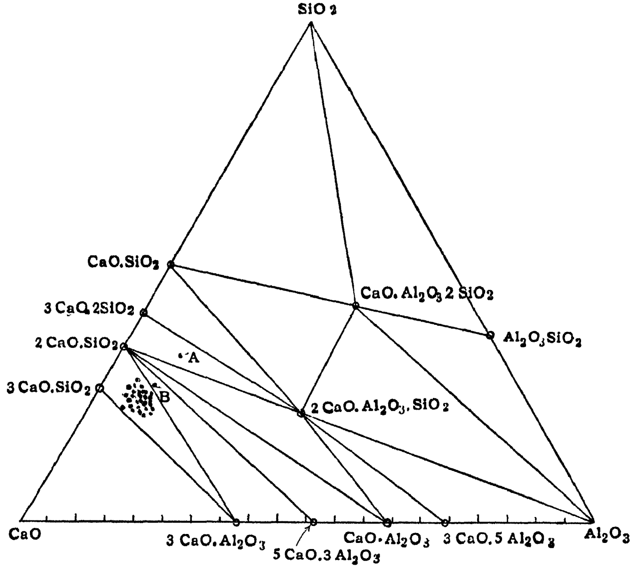

FIG. 1.—TRIANGULAR CONCENTRATION DIAGRAM SHOWING END PRODUCTS OF CONSOLIDATION. NORMAL PORTLAND CEMENTS ARE RESTRICTED TO THE TRIANGULAR AREA BOUNDED BY $2CaO.SiO_2$, $3CaO.SiO_2$ AND $3CaO.Al_2O_3$. (After Rankin and Wright, *op cit.*)

HYDRATION PRODUCTS OF PORTLAND CEMENT

When mixed with water, Portland cement undergoes the following changes:

1. The tricalcium silicate, $3CaO.SiO_2$, loses a molecule of lime, which at the same time hydrates; the remainder of the compound unites with the added water to form an amorphous, indefinite, hydrated silicate of calcium which slowly loses more calcium hydrate by hydrolysis. The calcium hydrate thus formed may be partly crystalline and partly amorphous, or even almost wholly crystalline, dependent on the amount of water used and subsequent conditions of wetness. When partly or wholly crystalline, it is distributed through the concrete, and especially between the sand and aggregate and the cement matrix both in extremely fine and in relatively coarse and large crystals.

2. Tricalcium aluminate, $3CaO.Al_2O_3$, when treated with water generates so much heat during its hydration as to expel some of the water

as steam; at the same time, so rapid is the rate of hydration that an amorphous envelope of hydrated material coats each particle and tends to prevent further and complete hydration. High alumina cements generally set rapidly, and are "hot" cements. As gypsum seems to retard the rate of hydration of this component, it is added to all Portland cement clinker before grinding. The hydrated product is largely amorphous, but may be in small part crystalline; the amorphous product delivers calcium hydrate by hydrolysis.

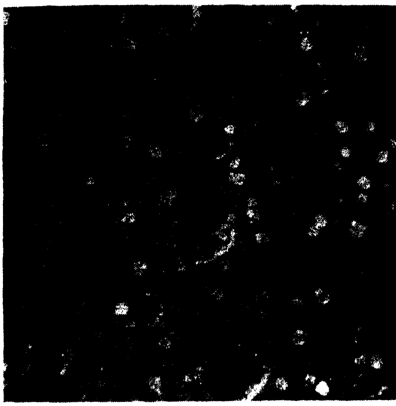

FIG. 2.

FIG. 3.

FIG. 2.—TAKEN IN ORDINARY REFLECTED LIGHT, SHOWING NUMEROUS HEXAGONAL CRYSTALS OF CALCIUM HYDRATE, WHICH FORMED ON BOTTOM OF TEST PAT OF NEAT CEMENT RESTING ON GLASS PLATE UNDER WATER. × 10.

FIG. 3.—TAKEN IN ORDINARY REFLECTED LIGHT. FIELD SHOWN IN FIG. 2 MORE HIGHLY MAGNIFIED, SHOWING MORE DISTINCTLY FORM AND CLEAVAGE OF CRYSTALS. × 40.

3. Tricalcium ferrite, $3CaO.Fe_2O_3$(?), hydrates to an indefinite, brown, lime-iron compound, quite amorphous, which delivers calcium hydrate by hydrolysis. The amount of calcium ferrite in ordinary Portland cements is never very large.

4. Beta calcium orthosilicate, $2CaO.SiO_2$, is the most difficultly hydratable component of cement. All of the unhydrated residues of cement clinker commonly seen (microscopically) in cement test pieces and in concrete consist of unhydrated granular aggregates of this component, which frequently enclose grains of the other components in such a way as to prevent them also from hydrating.

If the cement is ground sufficiently fine, beta calcium orthosilicate slowly takes up water of hydration and is converted to an amorphous indefinite hydrated calcium silicate that, by hydrolysis, yields more or less calcium hydroxide. Seldom is the action complete, and in the case

of rather coarse cements innumerable particles are wholly unhydrated, thus decreasing the efficiency of the cement.

5. Gypsum, $CaSO_4.2H_2O$, appears to control the rate of hydration of the tricalcium aluminate and the evolution of heat. Moreover, a chemical reaction between the gypsum and the hydrated aluminate is responsible for the development of silky needles of calcium sulfo-aluminate ($3CaO.Al_2O_3.3CaSO_4$ + much H_2O), a normal hydration product. Under some conditions this may become abnormal in size of crystals, quantity, and distribution, and one of the possible causes of disintegration of concrete

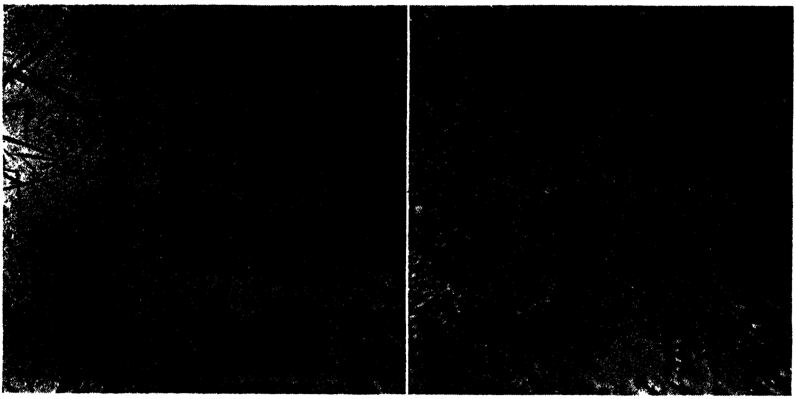

FIG. 4. FIG. 5.

FIG. 4.—TAKEN IN ORDINARY LIGHT. UNUSUALLY LARGE AND WELL DEVELOPED CRYSTALS OF CALCIUM SULFO-ALUMINATE FORMED IN WATER IN WHICH CEMENT TEST PATS WERE IMMERSED; THESE CRYSTALS ARE MUCH LARGER THAN THOSE USUALLY SEEN IN SET CEMENTS. × 90.

FIG. 5.—TAKEN IN ORDINARY LIGHT. CEMENT SHAKEN WITH WATER FOR A FEW MINUTES AND SETTLED; WATER SOLUTION THEN EVAPORATED. NOTE HEXAGONAL, PLATE-LIKE CRYSTALS OF CALCIUM HYDRATE. × 240.

in certain situations. Some of these features are illustrated by Figs. 2 to 7, which show the character of the hydration products mentioned.

RECASTING CEMENT ANALYSES

As the components of Portland cement have a definite composition, the various oxides found by chemical analysis must be uniformly distributed and united with one another according to fixed laws of chemical combination. Hence it should be possible to recast an analysis in much the same manner as rock analyses, and to obtain the percentage composition of the cement in terms of the actual compounds, instead of the oxides of the elements forming the mass. This is accomplished by dividing the percentages of the various oxides by their molecular weights, obtaining thus their molecular numbers, or ratios, and then allotting the

proper quantities of silica, alumina, lime, etc. to each in accordance with the various compounds that should form under the conditions of burning. Thus, sufficient lime is allotted to satisfy the requirements of CO_2, Fe_2O_3, SO_3, and Al_2O_3; the remaining lime is allotted to the silica to form both tricalcium silicate and calcium orthosilicate. This should be preceded, however, by a petrographic study of the cement in order to determine whether or not any abnormal components are present. For example, a cement analyzes as follows:

SiO_2, 23.40 per cent.; Fe_2O_3, 3.46 per cent.; Al_2O_3, 5.54 per cent.; CaO, 60.79 per cent.; MgO, 2.46 per cent.; H_2O, 2.59 per cent.; SO_3, 1.71

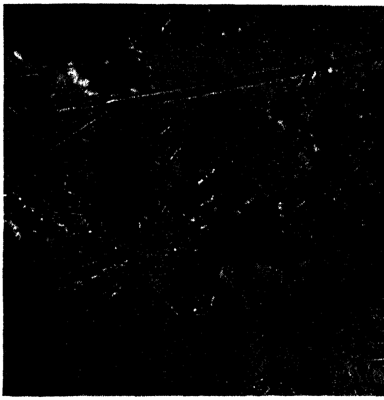

FIG. 6.

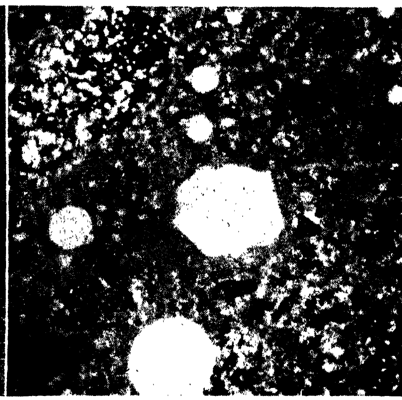

FIG. 7.

FIG. 6.—TAKEN IN ORDINARY LIGHT. CALCIUM SULFO-ALUMINATE IN NEEDLE-LIKE CRYSTALS, CALCIUM HYDRATE IN PLATE FORM (CENTER), AND AMORPHOUS HYDRATED CALCIUM SILICATE (DARK, SHAPELESS BLOTCHES). TAKEN FROM DEFECTIVE CONCRETE. × 350.

FIG. 7.—TAKEN IN ORDINARY LIGHT. THIN SECTION OF CEMENT TEST BRIQUETTE, HEXAGONAL PLATE OF CALCIUM HYDRATE IN CENTER. MANY UNHYDRATED RESIDUES OF CEMENT (BETA CALCIUM ORTHOSILICATE GRAINS), AND MUCH FINELY CRYSTALLINE AND AMORPHOUS CALCIUM HYDRATE. ROUND WHITE SPOTS ARE VOIDS LINED WITH CALCIUM HYDRATE IN MOST CASES. × 120.

per cent.; CO_2, 0.45 per cent. A "recast" of the analysis will then be as follows:

	Molecular Ratio	$CaO.CO_2$	$CaO.SO_3$	$3CaO.Fe_2O_3$	$3CaO.Al_2O_3$	$3CaO.SiO_2$	$2CaO.SiO_2$
SiO_2.......23.40 ÷ 60	0.390					0.049	0.341
Fe_2O_3...... 3.46 ÷ 160	0.021			0.021			
Al_2O_3...... 5.54 ÷ 102	0.054				0.054		
CaO.......60.79 ÷ 56	1.085	0.010	0.021	0.063	0.162	0.147	0.682
MgO....... 2.46				in solid solution			
SO_3........ 1.71 ÷ 80	0.021		0.021				
CO_2........ 0.45 ÷ 44	0.010	0.010					

The percentage composition, in terms of the components, may be found by multiplying the molecular numbers allotted (or component constants) by the molecular weights of the compounds; thus, the molecular weight of tricalcium silicate, $3CaO.SiO_2$, is 228, which, multiplied

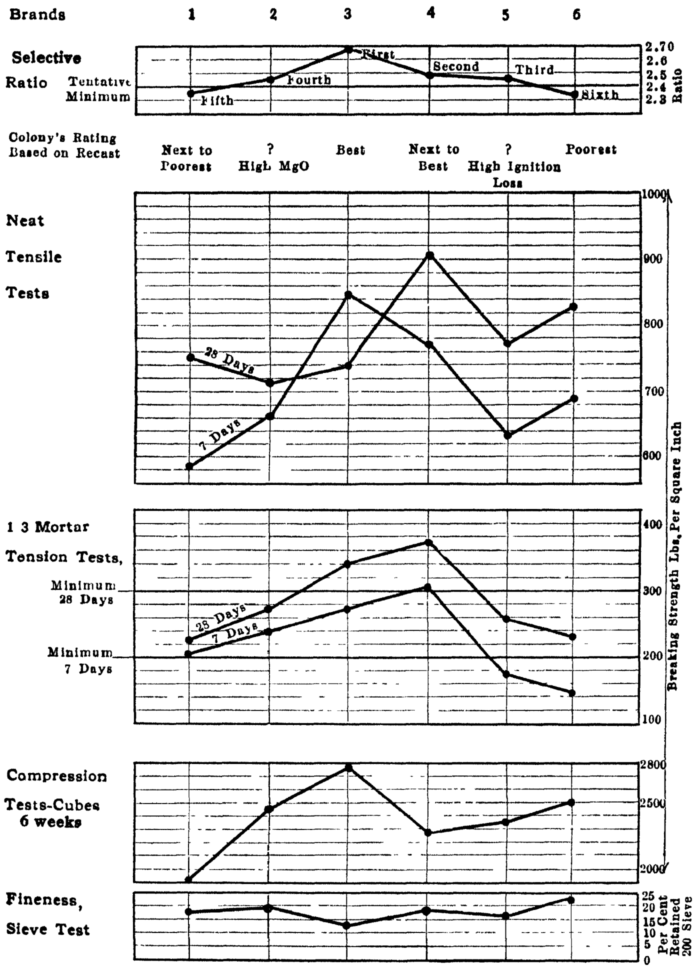

FIG. 8.

by the component constant (0.049) gives as a product 11.172, or 11.2 per cent. The componental composition of the cement is thus found to be: Tricalcium silicate, 11.2 per cent.; calcium orthosilicate, 58.7 per cent.; tricalcium aluminate, 14.6 per cent.; tricalcium ferrite, 6.9 per cent.;

gypsum, 3.6 per cent.; calcium carbonate, 1.0 per cent.; magnesia, MgO, 2.46 per cent.

It has been established by experiment[3] that the component of greatest cementing value is tricalcium silicate, and the component most difficultly hydratable is calcium orthosilicate. It follows, therefore, that the cementing value of any mixture of these components, such as Portland cement, is dependent on the proportions existing between the components, and the fineness of grinding of the mixture. In the example given, as the cement has only 11. 2 per cent. of tricalcium silicate (the best hydraulic component) and 58.7 per cent. of calcium orthosilicate (the most difficultly hydratable component), particularly if relatively coarse, it should be regarded with suspicion.

Importance of Fine Grinding

The fineness of grinding is an important feature and one not properly determined by the standard sieve test; although three-quarters of the cement may pass the 200-mesh sieve, the bulk of it may be, and frequently is, too coarse for efficient work, especially if it is high in orthosilicate and low in tricalcium silicate. When a relatively coarse cement of this character is used in concrete work, a petrographic study of thin sections cut from the concrete will prove the presence of a large amount of unhydrated cement clinker. In many cases from 25 to 40 per cent. of the cement is unhydrated or partly hydrated, resulting in a weak, permeable concrete mass that readily lends itself to attack by the agents of disintegration.

Limiting Ratios of Cement

By considering cements from the standpoint of their componental composition and selecting those with proper adjustments between these components, it is possible to guard against subsequent disintegration of completed structures to a very considerable extent. As an aid to proper selection, a limiting ratio based on a minimum tricalcium silicate content may be used. If the percentages of lime, silica, alumina, and ferric oxide, the sum of which in all normal Portland cements always exceeds 90 per cent., are converted to molecular ratios, the quotient obtained by dividing the molecular ratio of the lime by the sum of the molecular ratios of the alumina, silica, and ferric oxide, may be used as a selective ratio. The writer suggests 2.4 as the limiting ratio. All cements having a lower ratio than this should be regarded with suspicion.

[3] A. A. Klein and A. J. Phillips: Hydration of Portland Cement. *Tech. Paper* 43, Bureau of Standards (1914). See also P. H. Bates and A. A. Klein: Properties of the Calcium Silicates and Calcium Aluminate Occurring in Normal Portland Cement. *Tech. Paper* 78, Bureau of Standards (1917).

TABLE 1.—*Tests and Analyses*

	1	2	3	4	5	6
Standard Tests						
Normal consistency	22.5	22 75	23 0	22.75	23.0	21 0
Time of set, initial	4 hr 0 m.	5 hr. 30 m.	4 hr. 20 m.	4 hr. 0 m.	2 hr. 45 m.	5 hr. 45 m.
Time of set, final	8 hr. 5 m	8 hr. 10 m	6 hr 30 m	5 hr. 30 m.	6 hr. 0 m.	7 hr. 15 m.
Specific gravity	3 160	3 155	3 118	3 119	3.130	3 127
Soundness, normal	O K	O. K.	O K.	O. K.	O K.	O. K.
Soundness, accelerated	O K.	O. K.	O. K.	O. K.	O. K.	O. K.
Tensile strength, pounds per square inch:						
Neat, 7 days	581	669	859	776	635	690
Neat, 28 days	755	716	742	913	778	832
1 3 mortar, 7 days	203	240	273	306	175	150
1 · 3 mortar, 28 days	224	273	348	377	254	236
Per cent. retained on 200 mesh	20 0	21 4	17 0	19 8	17 6	22 4
Per cent retained on 100 mesh	1 0	1 4	4 0	2 0	2 2	1 6
Compression Tests						
2 in. cubes, 1 . 3 mortar, 6 weeks.	1920	2448	2756	2280	2380	2560
Chemical Analysis						
Silica, SiO_2, per cent.	23 79	22 61	19 88	21 43	22 14	23 40
Ferric oxide, Fe_2O_3, per cent.	2 64	2 51	3 97	2 90	3 17	3 46
Alumina, Al_2O_3, per cent.	4 86	6 00	6 73	6 80	5 47	5 54
Calcium oxide, CaO, per cent	61 31	61 37	63 48	62 14	61 80	60 79
Magnesia, MgO, per cent.	3 83	4 88	1 25	1 67	2 19	2 46
Water, H_2O (by loss), per cent	2 53	2 10	2 76	2 72	4 00	2 59
Sulfur trioxide, SO_3, per cent	1 45	1 41	1 75	1 79	1 24	1 71
Carbon dioxide, CO_2, per cent	0 48	0 33	0 57	0 59	0 59	0.45
Recast						
Calcium carbonate, per cent	1 1	0 8	1 3	1 3	1 3	1 0
Gypsum, per cent.	3 1	2 9	3 8	3 8	2 6	3 6
Tricalcium ferrite, per cent	5 2	4 9	7 9	5 9	6.6	6 9
Tricalcium aluminate, per cent	12 7	15 7	17 8	18 1	14 6	14 6
Tricalcium silicate, per cent	19 4	22 1	37 8	23 9	26 2	11 2
Calcium orthosilicate, per cent	53 5	48 2	28 4	43 3	43 7	58 7
Selective ratio	2 38	2 43	2 69	2 50	2 49	2.33

The use of the limiting ratio in this manner provides a simple, easy method of judging the quality of Portland cement, when used in connection with the usual standard tests; especially if the fineness of grinding is determined with the air analyzer. A more elaborate and careful study of the quality may be pursued by using petrographic methods in addition.

RESULTS OF LIMITING RATIO TESTS

As an example of the practical application of this method there are included in Table 1 and Fig. 8 tests and analyses of six standard brands of Portland cement. These were rated by the writer solely from the interpretation of the chemical analyses; the cements themselves were not seen, no micrometric measurements for fineness were made, and the results of the tests were unknown at the time of rating. On the basis of the recasts and the ratios cements Nos. 3 and 4 were judged to be the

best; Nos. 2 and 5 the next best; and Nos. 1 and 6 the poorest. The standard tests shown were made subsequently by the Testing Laboratory, Columbia University. The table and diagrams were compiled by the engineers for whom the ratings were made, and are used here by their permission.

To rate a cement solely from the chemical analysis, without having seen the cement, and without an opportunity for petrographic study is a severe handicap. Under the conditions, therefore, it is judged that the conclusions as to the quality of these six cements check very closely the results obtained by standard tests. The method contemplates the use of the air analyzer, microscopic inspection, and the interpretation of the chemical analyses; results obtained are so suggestive that further work is now in progress.

DISCUSSION

C. P. Berkey,* New York, N. Y.—Mr. Colony's work seems to indicate that it is going to be possible to determine more accurately than formerly whether a cement is good or poor. It is well known that the ordinary tests do not discover this fact in all cases early enough to be of much service, and many structures are in bad condition not so much because of poor workmanship as because a grade of cement wholly unsuited to the quality of work was used.

I have followed Mr. Colony's investigations for several years and I believe that he has proved that microscopic inspection of the original material aided by interpretation of a good chemical analysis brings out certain defects of quality with considerable certainty. It is one of the few cases in which the method of recasting analyses, such as is used extensively by petrographers, seems to have a practical application. It is evident, of course, that additional work ought to be done, especially along the lines of checking predictions with actual results under use and it is to be hoped that Mr. Colony will be able to continue his researches along this line. It is a promising lead and in view of the great economic importance of the problem it deserves encouragement from every quarter. It would be an immense relief to the engineer in charge of large construction projects if he could determine beyond question that the cement he is to use will give permanently satisfactory results.

Geo. A. Rankin, Washington, D. C. (written discussion).—The ideas advanced by Professor Colony should be of great value in the development of new methods for determining the soundness and strength of cements. A method that makes use of chemical, mechanical, and petrographical methods of analyzing cements should eliminate the difficulties

* Professor of Geology, Columbia University.

previously experienced. Thus, by determining chemically the various components of a cement; petrographically the chemical compounds that these components form during the burning of cement; and mechanically the fineness of the finished cement particles, it should be possible to predict the quality of a cement with certainty.

In his discussion of these methods Professor Colony, however, has neglected to use some of the more recent and reliable data concerning the constitution of Portland cement and the reactions that take place when cement is mixed with water and hardens. He gives tricalcium ferrite as the compound formed between lime and iron oxide; the work of R. B. Sosman and H. E. Merwin has shown[4] rather conclusively that it is the dicalcium ferrite and not the tricalcium ferrite that is formed.

In discussing the hydration products of Portland cement, Professor Colony states that tricalcium silicate loses a molecule of lime, which at the same time hydrates. Experiments carried out by the writer show that when tricalcium silicate is mixed with water it gradually loses lime, which forms calcium hydrate by hydrolysis, and if sufficient water is present the lime can be entirely separated from the tricalcium silicate. This results in a mixture of hydrated lime and silicate. At no stage in this dissolution is any hydrated calcium silicate compound formed.

In regard to the cementing qualities of the various constituents of cement, it would appear that the relative values given are open to question. The author states, "In the order of their cementing qualities these are: tricalcium silicate, tricalcium aluminate, tricalcium ferrite, and calcium orthosilicate, beta form." It is true that this is undoubtedly the order in which these constituents hydrate but that does not necessarily mean that it presents their relative value as cementing materials. Tricalcium aluminate hydrates very readily but it is of no particular value in developing the ultimate strength in cement. Calcium orthosilicate, on the other hand, while it is the most difficult to hydrate undoubtedly is a very important factor in the ultimate strength developed in cement and concrete structures, for, in hydrating, it liberates gelatinous silica, which gives to concrete its great strength. It is true that particles of calcium orthosilicate that are far from completely hydrated are often found in concrete or cement structures of long standing, but the calcium orthosilicate is nearly always partly hydrated and so provides some of the essential cementing material, gelatinous silica. This brings up the question as to whether or not it is best that particles of cement be completely hydrated in order to develop the maximum strength in concrete. This question has never been definitely settled, so far as the writer is aware, and would seem to depend on the relative strengths of the cement particles and those of the concrete aggregate. The available

[4] *Jnl.* Wash. Acad. Sci. (1916) 6.

data, however, would tend to show that if the particles of cement are only partly hydrated, being thus surrounded by a relatively thin coating of gelatinous silica, the ultimate strength of the cement or concrete is greater than if the available gelantinous silica be completely liberated.

Professor Colony states that ferric oxide will require three molecules of lime instead of two, as has been shown by Sosman and Merwin. So that in the example given, the percentage of tricalcium silicate actually present should be about 16 instead of the 11.2, and 55 per cent. of dicalcium silicate instead of 58.7. This correction is, however, a minor detail and does not seriously affect the deductions he has drawn from the recast analyses of cement.

It is to be hoped that this investigation will be continued, since it seems certain that it will be possible to develop tests which will determine the quality of cements with certainty only by carrying out the ideas he has outlined.

P. H. Bates, Washington, D. C. (written discussion).—This paper is disappointing for two reasons: First, after suggesting the use of the "componental composition" of a cement (or the percentage of the various constituents) for judging its quality, the author practically drops the method with but a minor discussion; and, second, after presenting the use of a ratio of certain of the oxides present, as shown by chemical analyses, he goes no farther than others who have approached the problem from this viewpoint, but considers it only from the strength developed at early ages by small specimens.

It should be borne in mind that ratios of this type, depending on the relative amounts of all or some of the five principal oxides (silica, iron oxide, alumina, lime, and magnesia) present in a cement are now used in proportioning raw mixes for cement burning the world over. In this country either the "old" or the "new Newberry ratio" is used.

$$\text{per cent CaO} = \frac{\text{per cent SiO}_2 \times 2.8}{\text{per cent Al}_2\text{O}_3 \times 1.1} \qquad \text{(Old)}$$

$$\text{per cent CaO} = \frac{\text{per cent SiO}_2 \times 2.5}{\text{per cent Al}_2\text{O}_3 \times 1.6} \qquad \text{(New)}$$

The Michael's ratio used in Germany, and well known as the hydraulic modulus, states that:

$$\frac{\text{per cent CaO}}{\text{per cent SiO}_2 + \text{per cent Al}_2\text{O}_3 + \text{per cent Fe}_2\text{O}_3}$$

should be between 1.8 and 2.2.

This latter ratio is that proposed by the author but not reduced to molecular values; in fact, long usage has not shown the necessity of reducing to molecular ratios.

The use of such ratios is more disappointing than the use of the ordinary small-specimen strength tests, as no one has shown the value of the former in predicting the properties of a cement in any of its applications at late periods of testing. A ratio of the type suggested places the high-silica cement in an unfavorable class, yet it is commonly accepted that such cements at late periods produce the most satifactory concrete. Is the fact or the ratio in error in this particular? Cement No. 6, cited by the author, has an unsatisfactory ratio, but in two of the five strength tests it ranks second highest. One of these two tests is the important mortar-compression test at 28 days. This has been suggested by Committee C-1[5] as possibly predicting the value of a cement in concrete more satisfactorily than the tension-test piece.

The author correctly quotes from the work of my associates and myself regarding the rate of hydration of the silicates of lime; but we were careful to state that while the orthosilicate hydrates slowly, it acquires, by the time of a 90-day test, a strength almost equal to that of the tricalcium silicate. We are therefore disappointed that the author did not include in his paper a discussion of the value of the suggested ratio or of the amounts of the various constituents present from the viewpoint of later testing than he has presented at this time. Not only must the question of a ratio or of the relative amounts of constituents be considered from the viewpoint of age but also from how the test piece was aged. The orthosilicate hydrates but slowly and requires time and continued moisture to hydrate properly; the tricalcium silicate hydrates so rapidly that it may have reached its maximum efficiency before the so-called excess water used in mixing has evaporated.

As the amount of hydration is a measure of the value of a cement, in the analysis of the ultimate value of a cement such factors as the composition or constitution can be considered of prime importance only in connection with the manner in which the cement is to be used. In mass concrete or in any concrete where excessive loads are not applied at early ages, and where it is subject to the presence of moisture, a cement high in silica or orthosilicate is not only permissible but desirable. A cement low in silica or high in tricalcium silicate would be desired for reinforced-concrete work.

To illustrate our point we are presenting in Table 2 the "percentage silica-alumina ratio" (a molecular ratio would change the values but does not rearrange the grouping) the average tricalcium and orthosilicate, and the average strength at certain periods of concrete made from about fifty cements. The cements were grouped according to selected

[5] Data Considered by Committee C-1 of American Society for Testing Materials in Preparing the Standard Specifications and Tests for Portland Cement. Published by Committee C-1.

silica-alumina ratios, as shown in the top horizontal column of the table, each cement being placed in that group nearest to which its ratio approached. The constituents were determined independently by two petrographers at different times, using the same thin sections of clinker. The content of tricalcium silicate, as found, was averaged for all the cements in any group, as was the sum of the tricalcium and orthosilicate. The latter was determined separately but not so presented, in order to condense the table. The compressive strength of a 1-1.5-4.5 gravel concrete made from the cements was averaged for all the cements in any group.

A study of the table immediately shows the value of the tricalcium silicate in producing early strength, but just as unmistakably shows the value of the orthosilicate at late periods. This latter is shown particularly in the two last vertical columns. These values only hold good for storage in a damp closet, they are materially changed by storage in water or the air in the laboratory. The silica-alumina ratio is not used here as being superior to the one used by the author. It is given solely because the entire table was prepared some time ago for another purpose. But it satisfactorily shows the need of studying all ratios and the amounts of constituents present from other viewpoints than the strength of small specimens at early ages.

TABLE 2.—*Effect of Di- and Tri-calcium Silicates on Strength, in Pounds per Square Inch, of Cements in a* 1-1.5-4.5 *Gravel Concrete at Different Ages. Arranged in Groups Depending on* SiO_2–Al_2O_3 *Ratio*

Average $\frac{SiO_2}{Al_2O_3}$	1 25	1 75	2 25	2 75	3 25	3 75	4 25	4 75
$3CaO.SiO_2$	43 0	25 4	33 4	28 7	18 2	34 9	25.2	8 3
$3CaO.SiO_2$ plus $2CaO.SiO_2$	62 5	64 2	71 0	75 8	78 3	80.4	82 3	82 3
Strength, at 4 wk.	1660	1205	1655	1315	960	1560	1560	570
Strength, at 1 5 yr	2375	2265	2700	3110	2670	3215	3240	2705
Strength, at 5 yr.	2650	3020	2960	3290	3170	3470	3610	3305

R. J. COLONY (author's reply to discussion).—The work of Sosman and Merwin on the system limeferric oxide, mentioned by Mr. Rankin, was not overlooked. When studying Portland cements and Portland-cement clinkers petrographically, however, the lime-iron compound was difficult to correlate with the very definite dicalcium ferrite described by those investigators, which possesses such well-defined optical properties. In Portland cements, the calcium ferrite (or ferrate?) is an indefinite, fuzzy, and generally amorphous, or apparently amorphous, substance which is

distributed interstitially with respect to the other grains, from which it was very difficult to derive any satisfactory results.

For this reason, the writer, tentatively, chose to regard it as related to the aluminate, and assigned to it the composition $3CaO.Fe_2O_3$, which may be incorrect, however, as Mr. Rankin suggests.

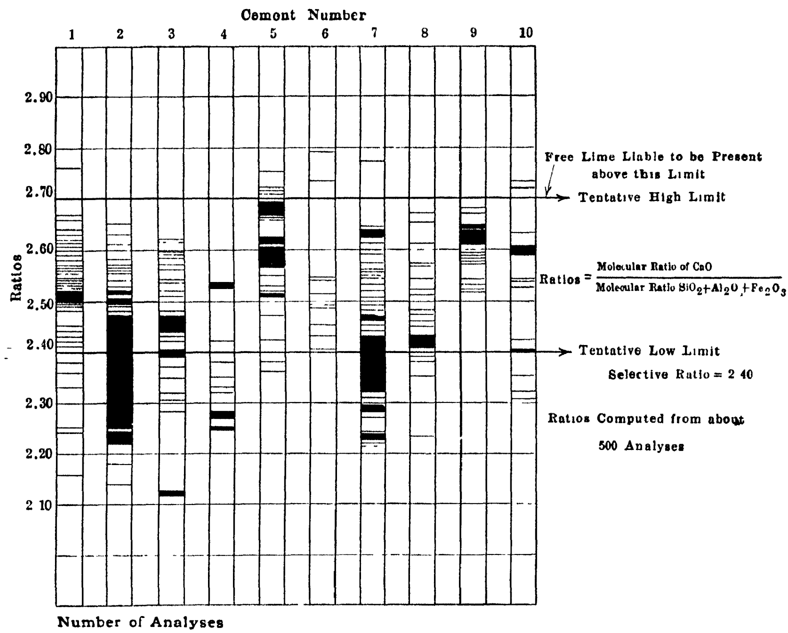

FIG. 9.

The comment of Mr. Bates, that the paper is disappointing because the use of the "componental composition" of a cement for judging its quality is merely suggested, without detailed discussion, is quite just. It was thought best not to discuss this method at the present stage of the investigation, however, deferring such discussion until additional experimental data are obtained, which may justify a better and fuller description.

The writer agrees with Mr. Bates, that ratios are inadequate as indices of quality. The selective ratio was devised merely as an aid in judging the quality of cements when the writer was working on cement problems for the Board of Water Supply of the City of New York. The ratio operates to condemn high-silica cements; it was precisely this type of cement that gave the most trouble on the aqueduct. Fig. 9 illustrates this point. Compiled from about 500 analyses and including ten brands

of cement, it shows that cements Nos. 2 and 7 were high-silica cements; that is, they contained, as the chief component, beta calcium orthosilicate. More trouble was experienced with these cements than with any others used on the aqueduct, notwithstanding the care used in testing and inspection.

The writer appreciates the fact that experiments show that, provided test pieces are kept long enough under ideal conditions in damp closets, calcium orthosilicate ultimately hydrates and gives, therefore, late strength to such test pieces, as shown by Mr. Bates, and by others. Unfortunately, however, the engineer cannot place his dam, or other structural unit, in a damp closet until such time as the orthosilicate hydrates and gives deferred additional strength to his structure. Natural processes of weathering, with which the geologist is so familiar, operate on artificial rock (concrete) as well as on natural rock, and these processes do not wait for the orthosilicate to hydrate; they start operation at once and proceed vigorously, in numerous cases with disastrous results to the structure.

It is the intention of the writer to continue this investigation, using the general principles outlined in the paper, and to definitely prove, or disprove, if possible, the ideas advanced.

Important Factors in Talc Milling Efficiency*

By Raymond B. Ladoo,† Washington, D. C.

(New York Meeting, February, 1921)

The milling of talc, as is the case with many non-metallic minerals, until recently, has not received adequate technical consideration, for the talc industry has become of importance only within the last decade. At first, talc was used only in the massive form, for foot warmers, griddles, and so on, and milling methods were unnecessary. As the demand for ground talc increased, producers adopted the machinery used in the milling of flour; in fact, many talc mills were rebuilt flour mills. Improvement has been slow, but today several types of grinding and separating machinery are in use; many of them, however, are still inadequate and inefficient.

To determine the best methods of milling talc, it is necessary to understand the essential properties a talc must possess to fit it for a particular use. For toilet purposes, whiteness, freedom from lime and grit, fineness of grain, and good "slip" are essential. As a paper filler or coating, the talc must be white, uniform, and fine grained, have a good slip, and be free from grit and iron; freedom from lime is a disputed point. Fibrous talc is supposed to be superior to massive on account of the interlocking of grains in the paper, thus increasing its strength; this point, though, is in doubt. In general, talc for the paper industry need not be of as high quality or as fine grained as that used for toilet purposes.

Talc is usually bought by sample, for which reason it is difficult to state the essential properties for use in paint, rubber, roofing, and so on. The adoption of standard screen tests and standard grades by producers is necessary before an accurate basis for manufacturing or selling standards can be established. Off-color talcs might be utilized, as in Germany and Austria, by the establishment of standard grades of colored talcs.

The machinery to be used depends on whether the talc is fibrous, foliated, or massive; hard or soft; of uniform or variable grade; pure or impure. In some cases, it is possible to change the mining practice or to utilize different sections of the deposit in order to vary some of these factors, but in many cases these factors are fixed, so the milling must be designed to suit the conditions.

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† Mineral Technologist, U. S. Bureau of Mines.

Processes and Machines now in Use

Crude talc from the mine, in many cases, is too moist and sticky to permit of efficient milling without first being dried. Sometimes it is dried in heaps exposed to the air; sometimes on a steam-heated, iron, drying floor; and sometimes in rotary, direct- or indirect- heat dryers fired with coal, coke, or steam. If direct-heat dryers are used, coke must be the fuel so that the color of the talc may not be impaired by smoke and soot. Where drying is necessary, rotary dryers are preferable.

Dryers may be essential in some mills; in others, they may be used only occasionally or they may be omitted entirely, especially if proper precautions are taken in mining.

The machinery for the primary reduction of talc rock does not differ essentially in various parts of the country. Crushing to about 1½ in. is usually done in jaw crushers of the Blake type, although for large production gyratory crushers might be used to advantage. Rolls or rotary crushers reduce these lumps to about ½ in. Between the primary and the secondary crushers there should be a rotary screen, which will remove material fine enough to pass the discharge opening of the secondary crusher.

The next step is accomplished by a variety of machines. The principal types used are the Raymond roller mill, the Fuller-Lehigh mill, various types of vertical and horizontal emery and burr mills, pulverizers of the swing-hammer and other types, so-called disintegrators, the Hardinge conical mill, and other continuous and intermittent types of ball and tube mills. These machines are followed by inclined vibrating screens, rotary screens, silk-bolting reels, or various systems of air separation; or the product may be bagged direct. Where screening or air separation is used, the product is conveyed to bins and then bagged. The product is carried from one stage of reduction to the next by bucket elevators, and belt or screw conveyors.

In the selection of all crushing, grinding, separation, and handling machinery care should be taken not only to provide sufficient capacity but some reserve; also that the capacities of the various machines are in proper proportion to one another. In one mill, the secondary crusher was too small for the rest of the plant, so that the mill capacity was lowered to the capacity of that crusher. Another mistake is to gage the capacity of a mill by the rated capacity of each machine running full time; it is impossible to maintain the maximum rated capacity for all machines simultaneously for any length of time.

The coarse and secondary crushing units are usually fairly efficient, but the full value of such machines is often not utilized. It is a fundamental principle that material already crushed fine enough should be removed before passing through the next reduction unit. Feed to a

crusher making a $1\frac{1}{2}$-in. product should be freed from all material finer than $1\frac{1}{2}$ in. by screening. If allowed to go through the crusher, it not only reduces the capacity of the crusher for doing useful work but it tends to cushion the crushing effect on the coarser material. When belt conveyors are used to feed crushers, a magnetic pulley at the discharge end is advisable. This removes all tramp iron, and not only protects the machines from injury but keeps out fine iron which may contaminate the finished product.

Opinions have differed widely as to the most efficient type of fine grinding and separating machinery, but certain fundamental principles should be recognized. First, the feed to a machine grinding to 200 mesh should not be over $\frac{3}{4}$ in., and $\frac{1}{2}$ to $\frac{3}{8}$ in. or finer is to be preferred. The very fine grinding of coarse material in one stage is not efficient, no matter what type of machine is used. Second, continuous operation is to be sought instead of intermittent, as in the case of certain types of pebble mills now in use.

Screening or bolting of material finer than 100 mesh is usually expensive, slow, and inefficient, though good results have occasionally been obtained, under careful competent supervision. Where dry grinding is used, some system of air separation is the cheapest and most efficient method of grading very finely ground talc. It has not been proved that fibrous talc from New York State, which contains fibers, flakes, and rounded grains, can be successfully separated by air, for, without fairly close sizing, large thin flakes settle in air at the same rate as small rounded grains. But here the problem may be solved by selective mining, close sizing before air separation, improvement in the design of air separation machinery, or the use of more than one separator in a closed circuit with the fine-grinding machinery.

Use of Ball Mills

The efficiency of ball or pebble mills for the fine grinding of talc is in dispute, but certain facts seem to be established. Pebble mills of the intermittent type, or "dump cylinders," are not economical; they are slow and have a small capacity. The finished product is not uniform and its grade is dependent on the opinion of the man in charge, as it is not mechanically controlled. As the grinding progresses, the inefficiency increases: the most active grinding is done in the first hour, for the fine talc gradually cushions the impact of the pebbles and little grinding is done at the end. In one mill 5000 lb. of pebbles are used to one ton of talc in a mill revolving for 5 to 7 hr. at 22 r.p.m. In roller mills, which are continuous in operation, equipped with air separation, when the talc is ground sufficiently fine it is automatically removed, so that the full crushing effect is continuously applied to the coarser material. Such

mills may have a capacity of about 5 tons per hour when producing a 200-mesh product.

Use of Tube Mills

Tube or pebble mills with continuous feed and discharge may be necessary in the milling of fibrous talc or talc mixed with tremolite; they may also be economical with certain other talcs, but the use of three or four large tube mills in tandem with no screening or separation between the mills nor for the sizing of the finished product is neither efficient nor economical. A good product is made by one talc producer by the use of short tube mills, but each mill is followed by bolting reels, which return the oversize for regrinding. In a plant being erected, a Hardinge mill and a large tube mill will be used with an air separation between them and another at the end of the tube mill; the coarse material from each air separator will be returned for regrinding. One plant practising close sizing between mills consumes 61 hp. per ton per hour; a plant that neither sizes nor separates between mills consumes over 375 hp. per ton per hour. Though there is some difference in hardness between the talcs ground in these cases, it is not enough to account for this difference in power consumption.

One argument given for the use of tube mills is that no iron produced by attrition of grinding surfaces can contaminate the product of a pebble mill lined with siliceous brick or pebbles. But iron can enter the pebble mill from the primary and secondary crushers. Furthermore, grinding machines in the most modern plants manufacturing the highest grades of toilet powders have all-iron grinding surfaces. It is stated that 0.5 per cent. of iron is not injurious in talc used for filling paper; that is, in milling a talc naturally free from iron, 10 lb. of metallic iron per ton of talc may be added without injury. In a mill producing 5 tons per hour, this would amount to 50 lb. of iron per hour, an amount of attrition that no mill could possibly produce and still be of practical use.

Vertical Roller Mills

Where a large capacity at uniform fineness is desired, a vertical roller mill is probably the most efficient for soft talcs. Such a machine, with automatic feed, in which the product is continuously removed by a current of air introduced from outside, is used in some of the best designed plants. For smaller capacities of fine material, swing-hammer and similar types of pulverizers and disintegrators and vertical emery mills equipped with air separation are used. Some types of these machines are suitable only for coarse material, 40 to 100 mesh, but when equipped with air separation, these mills may be used to produce a certain percentage of 200-mesh product. The tailings from the air separator must be carefully reground or sent to screens making a coarse

product. The tailings from all mills can be continuously returned for regrinding, but as there is a demand for coarser material, they are often sized and sold to the roofing industries. The best type of screen for this purpose is the flat, inclined, vibrating screen.

Horizontal Burr Mills

The inefficiency of horizontal burr mills, compared with vertical emery mills, for the grinding of most talc, seems to have been definitely established, as they have been discarded by nearly all producers.

Selection of Mill

The best type of hammer mill, pulverizer, disintegrator or emery mill for the grinding of a talc from a new deposit can be determined by a careful comparison of the physical properties of this talc with those of other talcs now successfully ground; or by actual tests made by companies selling grinding machinery. But whatever machines are used, the product from the last grinding machine should be carefully sized either by bolting or by air separation. No grinding machine can be relied on to produce continuously a fine uniform product without screening or air separation. Furthermore, the final, sized product should be tested frequently by hand screening. Probably nothing has injured the talc industry so much as the marketing of non-uniform, improperly sized product.

The finished products from all machines may be bagged by hand, but the cheapest and most efficient results are attained by the use of automatic bagging machines,[1] provided with ample bin capacity.

Grinding 300-mesh Material

With the Raymond, the Sturtevant, or similar air separation systems, a small amount of almost impalpable dust is produced. This is usually of very high grade but it cannot be obtained in large enough quantities to warrant its exploitation. No machine now on the market can produce by air separation such fine material in large quantities at a cost that will permit its sale at a reasonable price, for with increasingly fine grinding, the machine capacity is rapidly reduced, proportionately increasing the cost per ton of product. But one plant has found that a somewhat larger production of this grade may be obtained at only a slightly increased cost, by using a secondary system of dust collectors in conjunction with the primary air separation system. If the demand for very finely ground material, that is through 300 mesh, increases it may be necessary to use wet grinding and water classification.

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[1] Bureau of Mines, *Reports of Investigations* (August, 1919).

Electric Motors and Bins

Most of the older plants are steam driven, but there is a tendency to install electric drives in new mills and in the remodeling of old mills. Individual motor drives for each important machine have many advantages; in one mill, each motor has its own voltmeter, ammeter, and circuit breaker. This permits an accurate record to be kept of the performance of each machine and is valuable in checking efficiency and in estimating the cost of producing different grades of talc. A recording wattmeter would also be valuable.

A point in mill design often overlooked is that of providing ample bin capacity for crude talc, crushed rock, and for several grades of finished products. Most mills have adequate capacity for one of these, but rarely for all three. Bins are not expensive but they are of great value in insuring steady production when the mine is temporarily closed or when sections of the mill are shut down for a few hours. Among the objections to the use of bins for finished products are: that if through accident, improperly ground material is made a large quantity of good talc in the bin is contaminated; and that certain talcs when finely ground are very sticky and will not flow freely from a bin. The first objection may be removed by using a bin divided into compartments and frequently checking the product by screen tests. The use of several compartments is advantageous, in any circumstances, in order that several grades of product may be made. Frequent screen testing is likewise desirable in order that a high quality may be maintained; in several mills screen tests are made every hour. The second objection may be removed by heating the inside of the bins with exhaust steam, or by installing some system of mechanical stirring, or agitation. Finely ground material if dry and warm will often flow freely, when it will not flow at all if cold and slightly moist.

DISCUSSION

R. B. Ladoo.—There are no definite specifications for the selling of talc and I have not found a consumer who could tell exactly what he wanted. He tries a sample and if that is satisfactory he uses that talc. In fact, the tests are so poor that consumers have refused one kind of talc and accepted a talc made in just the same way, saying that it was satisfactory.

The first thing needed is to devise specifications. Certain consumers demand a talc with a slip, but no one can define or measure slip. There are no specifications whatever for those tests. So, first, the tests must be devised. The idea that fibrous talc must be used in the paper, paint, and other industries was probably originally fostered by some producers of that material. As a matter of fact, the fibrous talc is not so well fitted

for paper as the granular talc, for unless it is carefully prepared the fibers are apt to pierce the paper and make a bad spot. Less fibrous than granular talc is now used in paper. Some paint men object to fibrous talc because the fibers, if long, are apt to upend themselves in a film of paint, and be brushed off, leaving an opening. The talc men have not taught the consumers the uses of the various varieties of talc.

S. H. Dolbear, San Francisco, Calif.—To what extent is it possible to vary the amount of grit in the finished product by a selection of grinding machinery? Also, what results have been obtained with water flotation?

R. B. Ladoo.—The grinding test referred to has not been completed. All the screen tests have been made. The silica determinations have not been made, but from microscopic examinations of material remaining on the screens it seems probable that a method can be devised whereby probably 80 per cent. of the silica can be removed. The talc in question was foliated and when ground broke up into fine, thin scales, whereas the quartz was in little, rounded grains. I believe it is possible to grind that talc in either a Raymond roller mill, by which the coarser particles can be removed continuously from the inner cone, or by some form of a hammer mill equipped to throw out the heavier particles. I think that by one of these methods most of the grit can be eliminated.

If the talc is granular, I doubt if this could be done because the specific gravities of talc and silica or quartz are about the same, and when ground probably the grains would be about the same size. It would be difficult to separate grit or silica from talc in the case of a granular talc. Wet grinding and water flotation have not been practiced at all. Some experiments have been made, but no definite results have been obtained. I think that eventually the finer grades of talc, possibly the 300 mesh and finer, will have to be made by this method. I understand, however, that by air separation they are able to make a product of 350 mesh; but that would not apply to all talcs.

S. H. Dolbear.—If talc is fibrous it can undoubtedly be floated; we have succeeded in floating fibrous material, such as asbestos, with excellent results.

Relation of Gypsum Supplies to Mining

By D. H. Newland, Albany, N. Y.

(Wilkes-Barre Meeting, September, 1921)

Certain observations from the field and laboratory suggest the need for recasting some of our ideas about gypsum as a rock-forming mineral and in relation to supplies for industrial use. Until about 25 years ago, the use of gypsum was confined almost entirely to agriculture. During the past quarter century, though, its usefulness has been greatly developed, particularly in the building trades, which now absorb most of the gypsum produced. In 1919, according to the U. S. Geological Survey, the output of crude gypsum was 2,240,163 tons, a gain of 912 per cent. in 25 years. About one-half of this production was mined in New York, Ohio, and Michigan, where the proved supplies can hardly be regarded as over-large when viewed in relation to the rapid expansion of industrial requirements.

The occurrence of gypsum in the United States is treated by R. W. Stone and others.[1] The deposits of Canada are described by L. H. Cole.[2] Several states, including Kansas, Oklahoma, Iowa, Michigan, and New York, have made special surveys of their gypsum deposits, but the reports, as a rule, refer to conditions as they existed from 15 to 20 years ago.

Association of Gypsum and Anhydrite

The two calcium sulfate compounds, gypsum ($CaSO_4 \cdot 2H_2O$) and anhydrite ($CaSO_4$), are the more stable forms of a series which includes at least two other members, so-called half-hydrate ($CaSO_4 . \frac{1}{2}H_2O$) and soluble anhydrite ($CaSO_4$). Each of the latter readily takes up water to recrystallize as gypsum and consequently is incapable of existence under ordinary geological conditions; also each can be prepared from gypsum by the application of heat within certain temperatures, the process on which is based the manufacture of commercial plasters. If the temperature limitations are not observed, the result of the calcination process will be insoluble anhydrite which, compared with the soluble form, takes up water very slowly so as to be of no practical use, although actually it is soluble and in time is convertible into gypsum like the other forms.

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[1] U. S. Geol. Survey *Bull.* 697.

[2] Dept. of Mines, Canada. Ottawa, 1913.

The association of anhydrite with gypsum is rather common in deposits that have been mined or explored beyond the outcrop zone, that is to a depth of over 100 ft. (30 m.). A study of these occurrences would indicate that the form which calcium sulfate takes depends on environment; that is, the combination of physical and chemical conditions that pertains to any particular region above or below ground. There is nothing unusual about such relation in the occurrence of minerals in general, but its probable bearing on the question of gypsum supplies justifies some effort in the investigation of the details.

Van't Hoff and his associates,[3] in their experimental work on the minerals of the Prussian potash deposits, found that solutions of calcium sulfate, when evaporated in open containers, and therefore under atmospheric pressure, deposit gypsum or anhydrite according to the temperature reached at saturation. Up to 66° C. gypsum separates, above that limit anhydrite; however, if the solution contains other salts, the boundary temperature for gypsum-anhydrite will be lowered. In the presence of sodium chloride, anhydrite begins to form at 30° C. and the gypsum deposited below that temperature will, in contact with a solution saturated for sodium chloride, change into anhydrite. In the evaporation of sea water, the crystallizing point of anhydrite is 25° C.

From these data, it appears that the deposition of gypsum and anhydrite at atmospheric pressures is not simultaneous, but each substance crystallizes during a separate range of temperatures, which is lower for gypsum than for anhydrite. Therefore, in the evaporation of marine and saline lake waters, which undoubtedly were the sources of most of the calcium sulfate deposits, it would appear that the prevailing precipitate is likely to be anhydrite rather than gypsum, for such waters contain salts that must lower the boundary temperatures within the range of those ordinarily reached in dry climates.

It requires only moderate temperatures to expel the combined water of gypsum. Chemists are familiar with the fact that prolonged grinding in a mortar will drive off a considerable part, and it would be possible, undoubtedly, to make half-hydrate in a pebble mill with no other heat than that developed by friction of the apparatus, although perhaps not in a practical way. Observations in regard to the temperature necessary to effect complete expulsion of the water are somewhat conflicting. An unverified reference states that Shenstone and Cundall[4] dehydrated gypsum at 70° C.; Le Chatelier[5] places the required temperature at 163° C. Time

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[3] See especially Van't Hoff and Weigert: *Sitzungsber.* k. Akad. Wiss. Berlin (1901) 140.

[4] *Jnl.* Soc. Chem. Ind. (1907), **26**, 735.

[5] "Experimental Records on the Constitution of Hydraulic Mortars." Tr. by J. L. Mack. McGraw. N. Y., 1905.

and fineness of sample are factors that must be considered, and the time element cannot be satisfactorily gaged from laboratory experiments.

In view of the considerable change in volume (30 to 40 per cent. decrease) that accompanies the transformation, it is reasonable to suppose that pressure lowers the temperature of dehydration wherever there is opportunity for the released water to find escape. This is generally provided by the porous nature of the surrounding sediments, so that anhydrite may result under conditions like those represented, for example, by the burial of gypsum beds under a cover of sediments which provides a permanent load where the temperatures corresponding to the gradient of heat are perhaps but little higher than the averages at the surface. The latter conclusion cannot be confirmed directly by laboratory experiment; yet it appears well founded in principle and agrees with observations that have been made in the field. The inference to be drawn from the information at hand is that gypsum becomes unstable under conditions of moderate temperature and permanent load such as characterize all deeper deposits in the extensions of the outcrops.

On the other hand, anhydrite represents the unstable sulfate at atmospheric pressure and average surface temperatures. The prevalence of gypsum at the surface, its deposition by ground waters or in open cavities below the surface (veins), and the actual process of hydration which can be followed in samples of anhydrite exposed to the atmosphere, witness this fact. The change of anhydrite to gypsum in connection with some of the western deposits was described by A. F. Rogers,[6] who seems to have been the first to note the occurrence of the former as a rather common mineral in this country. The same transition has been observed by the writer in many deposits that have been explored in depth. The process of hydration has been in progress, no doubt, wherever anhydrite, by erosion, faulting, or other agency, has been brought to the surface or so near the surface that the pressure developed by the 50 per cent. or so increase in volume could find relief against the counter effects of load.

The record of an interesting experiment to determine the time required for gypsum to change into anhydrite came to the writer's attention while preparing this paper. The experiment originated with H. E. Kramm, who unfortunately did not live to complete it, and has been described by A. C. Gill.[7] About 50 gm. of natural anhydrite from Windsor, N. S., broken to pass a 6 or 8-mesh screen, were placed in a bottle, which was then filled with water and closed. Some lumps of dead-burned gypsum, or artificial anhydrite, prepared by heating gypsum from Hillsboro, N. B., for 3½ hr., to a maximum temperature of 560° C., were put into a second bottle, with water. Both containers were closed by a screw cap, but not

[6] *School of Mines Quarterly* (1915) **36**, 123–142.

[7] *Jnl.* Amer. Ceramic Soc. (Jan., 1918) **1**.

hermetically sealed. On examining the bottles six years later, it was found that the water had entirely evaporated and that the anhydrite had become an aggregate bound together by an intergrowth of gypsum crystals, which in some cases had attained a length of ½ in. The natural anhydrite had taken up 8.29 per cent. water, which shows that 34.15 per cent. gypsum was present. The dead-burned gypsum held 18.8 per cent. water, corresponding to a gypsum content of 87.46 per cent. On account of the evaporation of the water of the containers, the transformation process could not have gone on throughout the entire six years; consequently it must have been more rapid even than was indicated.

That anhydrite takes up water rather readily at the surface can be confirmed by observation of material exposed to the weather, as in the case of old mine dumps. In one such instance, anhydrite that had lain in the open for two years had accumulated a coating of gypsum that was readily discernible to the eye.

The change to gypsum is a process of solution and recrystallization. Its progress is affected by pressure, temperature, and presence of other salts in the solvent. Alkaline chlorides and sulfates assist solution, and it may be believed that saline waters like those that have been in contact with salt beds, which are not infrequently found in the same formations as gypsum and anhydrite, have been effective in the hydration under natural conditions.

Depth in Relation to Gypsum Beds

In view of differences of stability, it is to be expected that the field distribution of gypsum and anhydrites will conform to variations in environment and that, in general, their occurrence will not be coincident. While the transformation of anhydrite to gypsum, or the reverse, requires time, the period will be inconsiderable in most cases, compared with the long periods required to produce essential modifications of environment. In short, as a rule, the occurrence of gypsum and anhydrite should accord with their physical and chemical properties as related to existing geological conditions.

One of the more obvious relations that may be used as a guide to the geological distribution is depth from the surface, or better, perhaps, the vertical load; the two are not necessarily convertible terms. Most gypsum deposits are intercalated seams in a sedimentary series of shales and limestones or shales and sandstones. In the eastern districts and in most of those in the Middle West, the strata show little folding or faulting, but lie nearly horizontal or with a fairly regular dip in one direction. The load for such cases may be closely approximated from the depth, averaging around 115 lb. to the square inch for each 100 ft. of section. The presence of folds, fissures, or other structural irregularities will modify

the calculation, as happens in several districts where gypsum has been mined. This is also exemplified by the occurrence of gypsum in veins, which may exist at great depth, relatively, without involving any considerable load.

Under constant or slightly variant conditions of temperature and ground water, it is to be inferred that with increased load a limit is reached where gypsum is unstable and changes over into anhydrite. Similarly, anhydrite under relief of pressure will recrystallize as gypsum. Transformations of this character may have occurred repeatedly in connection with calcium sulfate deposits of the older geological formations from the time of their accumulation in open waters, so that it is impossible to trace the present beds through the different changes to the original precipitates.

There is considerable information at hand to show that the range of stability for gypsum under vertical load is so small that it will have to be given serious consideration in connection with the question of future mine supplies. The most circumstantial evidence comes from the mines in the East, where operations have been in progress the longest time. In western New York, where the deposits may be followed from the outcrop under an increasing cover in the direction of the dip, anhydrite begins to appear at 150 ft. (45 m.) or less and within the next 100 ft. becomes the predominant mineral. It is a fair inference that 250 to 300 ft. marks the limit beyond which gypsum cannot be expected in any considerable quantity, that is in its usual form of occurrence as seams under permanent load. Such gypsum as is found below that limit is the result of some special structural feature, to be considered on its merits. With reference to the deposits of Michigan, Grimsley[8] says that anhydrite is the prevalent form for depths below about 500 ft. In southern Ontario, the conditions are the same as for western New York. The relations in the Middle West are not so well known, as the only record obtainable is connected with the shaft at Centerville, Iowa, which was sunk in the last few years to open a gypsum deposit at the depth of 533 ft. As anhydrite occurs with the gypsum, or in separate bodies at least, the evidence is not altogether conclusive.

Drill records generally are not of much service, as they are likely to show all the calcium sulfate in the form of gypsum, on account of the similarity in appearance of the two minerals in a granular condition, like the drillings made by oil-well rigs, which have generally been used. It is quite probable that some of the reported discoveries of thick gypsum beds in the salt mounds of the Gulf coast at depths of 1000 ft. and more are to be explained on this ground. Error in identification may be held to be largely responsible for the impression, more or less general, that anhydrite is an uncommon mineral. It may be said that the domes of the Gulf

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[8] "Gypsum of Michigan and the Plaster Industry," 84. Geol. Survey of Michigan. Lansing, 1904.

region probably come within the class of structures that are an exception to the prevailing methods of occurrence where pressure is a direct factor of depth.

The writer is aware that certain gypsum districts offer difficulties in the application of the principle of a vertical arrangement of the two sulfate compounds, and that apparent anomalies may be found in many places. Such must be examined with the fullest attention to environmental conditions to uncover the real situation. R. C. Wallace,[9] in a paper on the Manitoba deposits, refers to a section in which 104 ft. of gypsum at the surface is succeeded by 25 ft. of anhydrite and this by 5 ft. of hard gypseous rock, and states that although the lower part of the upper gypsum shows evidences of having been formed from anhydrite the main body is probably in the same condition as it was when precipitated. The presence of gypsum below the anhydrite, on the other hand, is regarded as evidence that the anhydrite also is original. From the description, it would seem possible to explain the occurrence on the basis of an anhydrite bed that is undergoing change to gypsum but still incomplete in the lower zone. The lowermost bed of hard gypseous rock, also originally anhydrite, may have some physical characteristics that have assisted the change to gypsum before the upper bed was completely transformed. Such explanation is only suggested, as the present writer is unfamiliar with the district. The association of gypsum and anhydrite in Nova Scotia is also an apparent exception to the general rule and needs to be studied on its merits. It may be noted that the anhydrite in that region is generally found on steep slopes and cliffs, which probably have resulted from rather recent faulting, while gypsum is the prevailing mineral in deposits situated on flats and gently sloping ground.

On the basis of the evidence afforded by deposits that have been most fully explored, the conclusion seems fairly justified that the distribution of gypsum is rather closely controlled by environment and that pressure is an important factor in that connection. Observation in the Salina districts indicate that a depth of about 300 ft. (91 m.) is sufficient to hold up the transformation of anhydrite to gypsum. The evidence, admittedly, is not sufficient to give general effect to this deduction, but it is not improbable, at least, that the critical depth elsewhere will be found to be of moderate proportions.

Physical Features of Deposits

The common method of occurrence of calcium sulfate deposits is in seams or beds, to which the sulfate content undoubtedly is primary—an accumulation in the lake or marine basins in which the accompanying limestones, shales, or sandstones were laid down. But some deposits,

[9] *Geol. Mag.* [6] (1914) 1, 271.

though few in number and of minor importance for gypsum mining, have resulted by the reaction of sulfuric-acid waters on limestone in place, or from solution of the sulfate and its reprecipitation elsewhere.

Too much reliance may be put on the bedded appearance as an indication of the extent and uniformity of gypsum bodies. This appearance signifies little more than a broadly conformable arrangement with the enclosing strata and a development along the bedding planes more or less comparable to individual seams, say, of limestone or sandstone. As to the actual extent, outline or individual peculiarities of a deposit, only one guide can be safely used—exploration. Evidence taken solely from outcrops is likely to prove delusive, if used as a basis for estimating supplies.

Calcium sulfate exists in marine and like waters in small proportions. To account for the formation of beds 15 or 20 ft. thick, which are not uncommon, it is necessary to assume the evaporation of tremendous volumes of water, for 1 ft. of calcium sulfate represents the precipitated content of about 1400 ft. of sea water of the average salinity. Basins sufficiently deep to allow the evaporation of a quantity of water sufficient to precipitate the sulfate deposits in a single stage can hardly be regarded as within the realm of possibility. The explanation must lie along other lines. One factor that may have a bearing on their origin is that of re-concentration on the surface. The original deposits, thin and widespread, may have undergone a prolonged process of reworking on, or near, the surface with transfer of the sulfate by erosion and solution to local depressions in the basin, there to be reprecipitated. In that way, a substantial part of the sulfate within a wide area would be concentrated in beds of minor areal spread but of considerable thickness. This might be expected to bring about a reversal of the normal order of occurrence of the salt and calcium sulfate, which seems to be the usual condition, wherever the two materials are present in the same formation, as in New York, Michigan, Ohio, Kansas, etc.

At any rate no explanation of the origin of the deposits can be considered satisfactory that does not take into account the variations in the sequence, outlines, and makeup of the beds in different areas of the same district, a variability that is the rule rather than the exception.

In the semi-arid regions of the West, the deposits of gypsite (earthy gypsum) occupy small depressions in river flats, also the bottoms of lake basins, the waters of which have shrunk or disappeared. Red clay is the characteristic surface material and invariably surrounds the gypsite. Within river flats, the bodies of gypsite are seldom more than four or five acres in extent, but have the most varied outlines; there is little doubt that the depressions represent abandoned parts of the stream channel, since filled with gypsite. The calcium sulfate has been brought into place by percolating waters from the higher ground and from beneath the

beds, where they have come in contact with rock gypsum. The lake deposits lie in the lower central parts of the basin, which almost always has a rim of red beds. They are practically continuous throughout their extent, which may reach several square miles. A seasonal distribution of the rainfall seems an essential factor in the production of the deposits; a subsurface flow during wet periods takes place from the higher ground into the basin where it moves upwards by capillarity and under hydrostatic pressure to deposit the gypsum by evaporation, which proceeds rapidly in the dry season. At the bottom, the deposits are usually moist even in the driest weather.

Although there is no close analogy between such contemporary basin deposits and those formed under the environment of, say, Salina time from marine waters that had been barred off from the sea, they supply an interesting illustration of the relative rapidity with which calcium sulfate undergoes solution and concentration at the surface.

Single deposits in New York and Ohio have been well enough explored to indicate the general outlines. Their areas are measurable in terms of acres, or at most a few square miles, and they are incomparably smaller than the areas occupied by the formations that contain them. Each deposit is a unit by itself, with its own characteristics and physical conformation. To what degree these individual features reflect conditions of original environment it is difficult to determine in view of the long periods of time that the deposits have been under influences that make for change and the facility with which both gypsum and anhydrite yield to them. It is a case for special study, and no general statement would be of much value.

Features Resulting from Solutions

Gypsum and anhydrite are among the most soluble materials that ordinarily appear in outcrop. Their solubility is considerably less in pure water than in water containing other salts, like alkaline chlorides. It amounts, for gypsum, to 1 part in 378 parts water, and for anhydrite, to 1 part in 479 parts water, at a temperature of 24° C. Although, abstractly, this implies a condition of relative insolubility, from a geological standpoint it means a fairly soluble rock, which succumbs to meteoric waters more easily than limestone and is only exceeded in that respect by rock salt among the common minerals.

The activity of ground waters in taking up calcium sulfate is observable everywhere that deposits are within their reach, for all springs and wells in gypsum districts are heavily mineralized. Such "gyp" waters are sometimes useful guides to the prospector in areas where gypsum does not appear in outcrop. From the standpoint of industry and household supply, they are distinctly obnoxious, although it is possible after treatment to use them in boilers. In certain districts, bodies of pure

water exist below the shallower circulations that carry gypsum, and such may be developed by artesian wells.

Solution of the exposed edges of gypsum deposits leads to irregular outcrops, marked by wavy, indented, or dissected bodies separated by barren intervals, with settlement of the roof, and in a further stage to the disappearance of the whole outcrop. In thin seams, like those found in the Akron and Oakfield districts of western New York, the undiminished gypsum is first encountered under cover of 20 to 40 ft. of rock, although adjacent limestone beds show little effects of solution. The occurrence of gypsum near the surface produces peculiar drainage features through settlement and caving, which go on more rapidly than the progress of stream adjustment.

Outcrops by themselves are not dependable as evidence of conditions in the deeper zone. Overestimation of their importance has been one of the chief hazards of gypsum mining, for which statement it is possible to find substantial basis in almost all of the older fields. They have also been given undue value, probably, in calculations of geologists in regard to the resources of some of the important districts.

Summary

The association of anhydrite and gypsum is a common feature in districts where exploration has reached beyond the superficial zone.

Comparison of the physical and chemical constants of the two minerals indicates different degrees of stability under conditions represented by natural environment. Their distribution is not generally coincident, but is regulated by conditions of temperature, pressure, and solvent waters.

Gypsum is the stable form under ordinary atmospheric temperatures and pressure. It has a small vertical range of occurrence. In those districts where exploration has supplied reliable information as to the depth of the gypsum zone, the limit seems to be reached at not over 500 ft. and in one or two districts at not more than 250–300 ft. vertically

Anhydrite succeeds gypsum in depth where pressure contributes to the stability of the mineral with the more compact molecular arrangement. In the case of flat seams that support the full load of rock cover corresponding to the depth, estimated at 115 lb. to each 100 ft. of cover, the process of hydration is held up at the indicated limits and for any further increase of load, anhydrite is the permanently stable form. The change from the anhydrous to hydrous form calls for a volume increase of around 50 per cent.

The physical structure and characteristics of calcium sulfate deposits are considered with reference to their bearing on mine exploration. Bedding is not necessarily an indication of regularity or continuity.

Deposits are local accumulations and each district should be treated on its merits as an independent unit. Exploration by core drilling is the most satisfactory method of estimating conditions ahead of mining. Surface observations are conjectural and often unreliable.

Solution by ground waters is an important factor in producing secondary modifications of beds. Outcrops may fail entirely if the beds are thin. Wastage at the surface tends to shorten the range of gypsum occurrence, and mining complications are brought about by differential solution effects that are evidenced by mud seams, caves, and sinks. Gypsum ranks as the most soluble of the minerals that commonly appear in outcrop.

Veins and cavity deposits constitute an exception to the prevailing order of arrangement, and this is possibly true of the sulfate occurrences represented by the Gulf coast mounds, inasmuch as their structure implies a relief of load that admits the existence of gypsum beyond the usual depth for a loaded seam.

Altogether, gypsum represents a somewhat special field of mining geology wherein exists a need for systematic study, which is likely to be appreciated more and more with the continued progress of the industry.

Application of Ball-mills in Southeast Missouri

BY LEWIS A. DELANO,* E. M., BONNE TERRE, MO., AND HAROLD RABLING,† B. M. E., LEADWOOD, MO.

(Lake Superior Meeting, August, 1920)

IT HAS been generally recognized that, owing to the extreme friability of galena, fine grinding has a tendency to cause excessive sliming of the mineral, so operators of lead mills have attempted to avoid this condition by commencing concentration at as coarse sizes as any mineral is freed. Jigging has been the basis of the milling process, making a clean coarse concentrate and a large amount of tailing, which was considered sufficiently low-grade to be rejected. However, developments and improvements in concentration processes are everywhere changing the low-grade tailing of yesterday into material worth re-treating today. Investigations of the work of the Hancock jig on the Bonne Terre ore have shown that the best results to be expected will give a tailing assaying 0.75 per cent. lead; the average for all conditions and grades of ore will be between 0.80 and 0.90 per cent. By crushing the jig tailings in ball-mills and retreating by tables and flotation, a tailing of considerably lower grade can be made; the problem then becomes an economic one of increased crushing costs versus higher recovery. In considering the entire elimination of jigging, the increased smelting charges due to a greater amount of slime galena will be counterbalanced by the lower operating cost, increased recovery, and the many advantages of a simpler flow sheet.

In developing these problems in the Bonne Terre mill of the St. Joseph Lead Co., a long series of tests has been carried out covering every stage of crushing and concentration, and a large number of possible flow sheets were considered and tried. Only the final results and a few comparisons will be given here. At the same time, a careful and detailed cost-keeping system has enabled us to compare milling costs with the various flow sheets, and to calculate the economics of the problem with great accuracy.

CONCENTRATION PROBLEM

The ore occurs as widely scattered deposits of galena disseminated through limestone; it has a specific gravity of 2.8 and the following composition: Pb, 4.00 per cent.; Fe, 3.00 per cent.; Zn, 0.32 per cent.; S, 3.10 per cent.; MnO_2, 0.73 per cent.; SiO_2, 4.20 per cent.; Al_2O_3, 3.20 per cent ; CaO, 28.60 per cent.; MgO, 16.85 per cent.; CO_2, 36.00 per

* Mill superintendent, Bonne Terre Mill, St. Joseph Lead Co.
† Mill superintendent, Leadwood Mill, St. Joseph Lead Co.

cent., by difference. Pyrite and marcasite are associated with the galena, also sphalerite and chalcopyrite, but in quantities too small to be worth recovering.

As the ore is low grade, the handling of large tonnages with a low operating cost is essential to the maintenance of the industry, and concentration is of vital importance. Approximately 20,000 tons of 4-per cent. ore are concentrated in the district daily, in seven mills treating from 2000 to 4000 tons each, and methods have been practically identical throughout. The ore is crushed through 9-mm. or 10-mm. (round hole) screens before any attempt is made at concentration. It is then separated over 2-mm. (round-hole) screens; the oversize forms feed for the Hancock jigs and the undersize is deslimed for table feed, the slime going to the flotation plant. Briefly then, there are three stages of concentration, namely, jigging on material between 9 mm. and 2 mm., tabling on material between 2 mm. and 150 mesh, and flotation on slimes, each stage forming a final concentrate and tailing. The middlings are recrushed in rolls and returned to the original circuit. The development described in this paper has eliminated the coarse concentration and simplified the process to tables and flotation only.

In Table 1 is shown a screen analysis of the Bonne Terre ore as it is delivered from the primary crushing plant, that is, the undersize of the 9-mm. screens, which is the point at which concentration is commenced. The last column was obtained by sorting each size and represents the

TABLE 1.—*Screen Analysis of 9-mm. Undersize*

Mesh	Per Cent Weight	Assay Per Cent. Lead	Lead Content, Pounds per 100 lb. of Ore	Per Cent. of Lead Content as Free Galena
On 3	5.6	1.66	0.093	
4	17.3	2.66	0.460	5.0
6	14.3	2.94	0.420	10.5
8	11.4	3.40	0.388	22.5
10	8.7	3.74	0.334	36.2
14	6.2	4.12	0.255	53.6
20	4.9	4.48	0.220	63.8
28	5.0	5.18	0.260	73.0
35	3.3	5.88	0.194	80.4
48	2.6	6.70	0.174	85.8
65	1.9	8.06	0.153	90.2
100	2.1	8.26	0.174	93.5
150	1.5	7.30	0.110	97.0
200	2.7	6.76	0.183	100.0
Through 200	12.5	7.48	0.937	100.0
Totals.......	100.0	4.35	4.355	61.3

percentage of the lead content that is freed sufficiently to give a concentrate of 70 to 75 per cent. grade. It will be noticed that in the coarser sizes only a small proportion is freed. This is shown more clearly in Fig. 1, which shows the cumulative free lead in the oversize and undersize of any mesh screen. For example, by screening through a 10-mesh

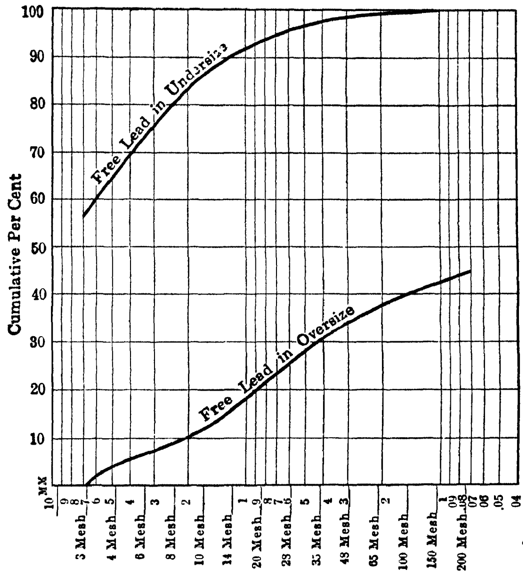


FIG. 1.—FREE LEAD IN ORE AT ANY MESH.

screen, the lower curve shows that 12 per cent. of the lead in the oversize is free; the upper curve shows that 86 per cent. of the lead in the undersize is free. The preliminary separation by screens and deslimers at 2 mm. and at 150 mesh gives three products, which are shown in detail in Table 2.

TABLE 2.—*Preliminary Separation of Crushed Ore*

Material	Per Cent. Weight	Assay Per Cent Lead	Lead Content Pounds per 100 lb. of Ore	Per Cent of Lead as Free Galena	Pounds Free Lead per 100 lb. of Ore	Distribution of Free Lead Per Cent.
Jig feed, 9mm. to 2 mm.	66.3	2.56	1 700	15 40	0 262	9.8
Table feed 2mm. to 150 mesh	18.5	10.30	1.912	87.17	1.670	62.4
Flotation feed through 150 mesh	15.2	4.80	0.743	100 00	0.743	27.8
Total ore	100.0	4.35	4.355	61.30	2.675	100.0

Of each 100 tons of ore, 66.3 tons go to the jig, 18.5 tons to the tables and 15.2 tons to the flotation plant. As considerable middlings are reground, the total feed to all machines is higher than this, but these figures are a good basis for comparison. The low percentage of free lead in the jig feed is noticeable; according to Table 2, of 2.675 lb. (1.21 kg.) of free lead in the total ore, only 0.262 lb. (0.12 kg.) (or 9.8 per cent.) is in the jig feed. Assuming that the lead recovered from recrushed middlings is evenly distributed over the three concentrating stages, it is reasonable to suppose that the production of concentrates will be practically in the ratio shown in the last column of Table 2. Actual tests check these figures very closely, and prove that as a producer of concentrates the jig is of minor importance. As a producer of tailings, however, the jig has been of primary importance, rejecting over 50 per cent. of the ore as tailings in the coarse sizes and enabling the mill to handle large tonnages at low cost.

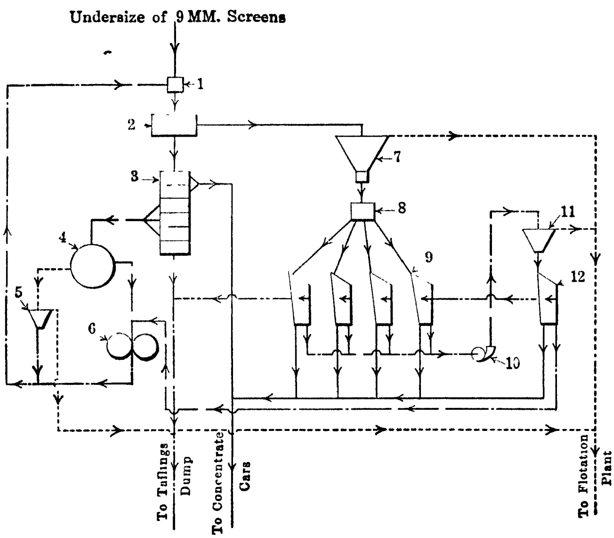


FIG. 2.—OUTLINE OF FLOW SHEET BEFORE BALL-MILLS WERE INSTALLED.

1. Elevator
2. 2-mm. screens
3. Hancock jig
4. Dewatering wheel
5. Settling tank
6. Rolls
7. Deslimer
8. Distributor
9. Wilfley-Butchart tables
10. Pump
11. Dewatering box
12. Wilfley-Butchart table

Table 3 shows an average of two months' regular operation in the Bonne Terre mill in 1916, before ball-mill crushing was introduced, and with the flow sheet shown in Fig. 2. It is typical of the operation of the whole district, and is the basis from which the experiments were begun to determine the application of fine crushing.

TABLE 3.—*Results of Operation Without Ball-mills*

Material		Tons per 24 Hr	Assay Per Cent. Lead	Lead Content Tons per 24 Hr.	Distribution of Lead Content in Final Products Per Cent.
Ore		2060 00	4.00	82.400	100.00
Concentrates	Jig	15.30	70.00	10 710	13.00
	Table	58.67	70.00	41.073	49.84
	Flotation	28.64	60.00	17.184	20.85
	Total	102 61	67.20	68 967	83.69
Tailings	Jig	1060 00	0.85	9 010	10 30
	Table	565 23	0 60	3.427	4.80
	Flotation	332.16	0.30	0 996	1 21
	Total	1957 39	0 69	13 433	16 31

CRUSHING TESTS

In July, 1916, a 6 ft. by 22 in. Hardinge conical mill was installed in No. 1 section for test purposes and experiments were commenced to determine the conditions for most effective work. Later a 6 by 4 ft.

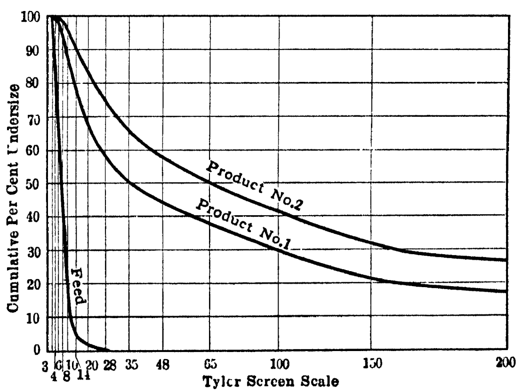


FIG. 3.—GATES CRUSHING DIAGRAM FOR HARDINGE-MILL TESTS.

Allis-Chalmers ball granulator was installed in No. 3 section, and a series of comparative tests were made. The results led to the adoption of the Allis-Chalmers mill, from which the grates had been removed, making it an open-trunnion mill.

It should be understood that we are not advocating any ball-mill in preference to others. The fact that the Allis-Chalmers open-end mill was adopted does not prove that it showed a greater efficiency, based on the

laws of crushing, but that it was found most suitable for our particular purposes. As a matter of fact, crushing efficiency was the last thing considered, as it did not give a true indication of the most desirable mill. Fig. 1, showing the amount of lead freed at various mesh sizes, indicates that extremely fine crushing is not required on this ore; 96 per cent. of the lead is freed when the ore is crushed to pass through a 28-mesh screen, and practically 100 per cent. when crushed to pass through a 100-mesh screen. A high percentage of slime in the ball-mill product is not only unnecessary but is extremely undesirable because of the larger amount of lead going to the flotation plant. On the other hand, a ball-mill operated so as to give a low percentage of slime will show poor crushing efficiency. Table 4 shows the results of two tests with the Hardinge mill operating under different conditions of moisture and tonnage rate of feed. The

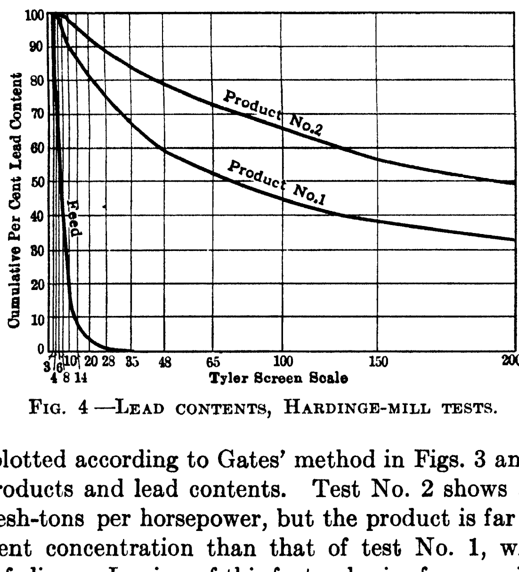


FIG. 4.—LEAD CONTENTS, HARDINGE-MILL TESTS.

results are plotted according to Gates' method in Figs. 3 and 4, showing curves of products and lead contents. Test No. 2 shows a higher efficiency in mesh-tons per horsepower, but the product is far less desirable for subsequent concentration than that of test No. 1, with its lower percentage of slime. In view of this fact, a basis of comparison has been worked out to give preference to the mill and to the operating conditions which show the highest capacity in tons crushed through a 10-mesh screen, the highest ratio of tons through a 10-mesh screen per horsepower, and the lowest percentage of minus 200-mesh material in the finished product. On this basis, the comparison of the two tests shown in Table 5 proves that test No. 1 gave better results in every respect.

Throughout the crushing tests, the size of feed was constant, being the material through 9-mm. and on 2-mm. (round-hole) screens. The screen analysis shown in Table 4 is a good average sample of the material crushed.

TABLE 4.—*Hardinge Mill Tests With Different Proportions of Water*

				Test No. 1			Test No. 2		
Tons feed per day....	....			240 0			190.0		
Per cent. water.....	....			65 8			48.1		
Horsepower..........	....			51 6			51 6		
Mesh-tons per horsepower...........	....			535 0			567.0		
	Average feed			Product			Product		
Mesh	Per Cent. Weight	Assay Per Cent Lead	Per Cent. Lead Content	Per Cent Weight	Assay Per Cent Lead	Per Cent. Lead Content	Per Cent Weight	Assay Per Cent. Lead	Per Cent. Lead Content
On 3	3.1	1 44	2.8						
4	11 9	2 12	15 7	1 0	1.06	0 7			
6	17 6	1 82	20 1	1.7	1.06	1.1	0 3	0 94	0.2
8	26 0	1.34	21 9	3 8	1.00	2 4	1.0	0 42	0 2
10	24 7	1 36	21 1	7 8	0.92	4 5	3.5	0 76	1.7
14	12 4	1 40	10 8	8 8	0 86	4 8	5 2	0 79	2 6
20	4 3	2 84	7 6	9.0	1.02	5 8	7.0	0.59	2 5
28	....	...	.....	9 7	1 04	6.4	9.3	0 74	4.3
35	.....	....	.....	7.8	1.32	6 5	8 3	0.79	4 1
48	....	....	.....	6.7	1 78	7.6	7 9	1.16	5 7
65	....	....	.....	6.0	1 64	6.1	7.1	1.23	5 4
100	......	....	....	8 3	1 60	8 3	8 0	1.43	7.1
150	.....	...	....	8 0	1.24	6 2	10.6	1 48	9.7
200	.....	....	.....	3 9	2 28	5 7	5.3	2.10	6.9
Through 200	....	...	.....	17 5	3 08	33 9	26.5	3 00	49.6
Totals	100 0	1 60	100 0	100.0	1 60	100 0	100 0	1 60	100 0

TABLE 5.—*Hardinge Mill Tests*

	TEST 1	TEST 2
Tons feed per day....................	240 0	190.0
Per cent. water..........................	65.8	48.1
Horsepower..............................	51.6	51.6
Capacity, tons through 10 mesh	216 0	181.0
Tons through 10 mesh per horsepower....................	4 18	3 51
Per cent. slime in 10-mesh undersize	23 8	33.4
Per cent. lead content of slime.	43.4	57.7

The ball-mills were operated in open circuit in every case, and tests were made, as far as possible, under normal operating conditions. Over 200 tests were made in all, but only the final results and some general conclusions will be given.

FACTORS AFFECTING CRUSHING

The capacity of the ball-mills and the nature of the product were considered to be governed chiefly by the following factors: Speed of rotation, weight of ball charge, nature and size of balls, moisture in feed, rate of feed.

Speed and Ball Load

The speed and ball load directly determine the power consumed. Within certain limits, variations in speed did not give a marked difference in the work done, but the tests in this direction have not been as extensive, nor the results as conclusive, as we would wish. Apparently, not much variation is experienced between the limits of 420 to 490 ft. per min. peripheral speed for the cylindrical mill, and between 450 and 550 ft. per min. for the Hardinge.

Variations in the ball load also have been found to be of comparatively minor importance. A heavy ball load consumes more power, but gives higher capacity, up to the point where the balls are discharging through the trunnion, and the amount of crushing done per horsepower remains practically the same. Table 6 shows the combinations of speed and ball load that have been found to give the best results.

TABLE 6.—*Speeds and Ball Charges*

	6 ft. by 22 in. Hardinge Mill	6 by 4 ft. Allis-Chalmers Peripheral-discharge	6 by 4 ft. Allis-Chalmers, Open Trunnion
Speed, revolutions per minute..	28	23	26
Weight of balls, pounds........	10,000	12,000	12,000
Horsepower..................	51.6	62.5	69.0

Size of Balls

The size of balls is important in determining the nature of the product. Balls of 5-in. diameter are necessary to crush the coarse ore; small balls increase the slime in the product. During the earlier tests, when cast-iron balls were used, considerable trouble was experienced by the breaking up of the balls, until the charge consisted largely of very small pieces of 1 in. diameter or less. Samples taken from day to day during this period showed a gradual increase of oversize and in the amount of slime. The cast-iron balls were therefore replaced by steel balls from 5 to 2 in. in diameter. Various combinations of these sizes have been used, and the highest capacity and lowest slime have been obtained with a charge containing a large proportion of 5-in. balls. The charge now used is calculated to give an equal area in each size, a 12,000-lb. charge being made up of 5460 lb. 5-in. balls, 4040 lb. 4-in. balls, and 2500 lb. of 2½-in. balls. New balls to make up for normal consumption are added weekly in the same ratio of sizes as the original charge; this practice has given satisfactory results.

TABLE 7.—*Ball Consumption Data*

Mill	Hardinge Mill			Peripheral-discharge Mill	Open Trunnion Mill
Material	Cast Iron	Steel	Steel	Steel	Steel
Shifts operated	830	2,254	327	1,635	1,908
Ball consumption, pounds	68,585	50,500	3,370	22,312	25,521
Tons ore crushed	56,280	151,883	28,100	149,444	198,428
Consumption, pounds per ton	1.218	0.332	0.120	0.150	0.143
Ball charge, in pounds	2-in., 4,500	2½-in., 6,000	5-in., 4,700	5-in., 500	5-in., 500
	1½-in., 3,300	2-in., 4,000	4-in., 2,950	4-in., 1,500	4-in., 1,500
	1¼-in., 2,200	1½-in., 2,000	2½-in., 2,350	2½-in., 7,000	2½-in., 7,000

Moisture Content

The moisture content of the material crushed affects the results mainly in the very fine sizes. A low moisture content increases largely the amount of slime in the product, without materially decreasing the amount of 10-mesh oversize. This is shown in Figs. 5 and 6, where capacity and per

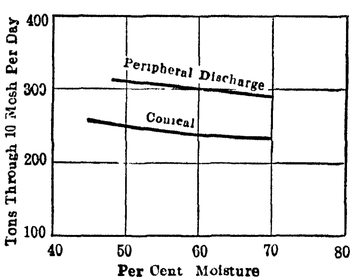


FIG. 5.—EFFECT OF MOISTURE ON CAPACITY.

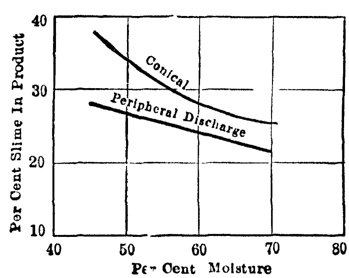


FIG. 6.—EFFECT OF MOISTURE ON SLIME.

cent. slime are plotted against the per cent. moisture, for a series of tests on the Hardinge mill at 300 tons of feed per day, and the Allis-Chalmers mill at 375 tons of feed per day. By increasing the moisture from 45 to 70 per cent., the capacity of the Hardinge mill drops from 258 to 232 tons, which is approximately a 10 per cent. decrease, while the slime drops from 37.8 per cent. to 25.5 per cent. a decrease of 33 per cent. In the case of the peripheral-discharge mill, the difference is not so marked, but in all cases it has been found better to sacrifice a little capacity in order to obtain a product with low slime content. An average of 60 per cent. moisture is, therefore, maintained in regular operation.

Rate of Feed

The effect of rate of feed is shown in Figs. 7 and 8, where the capacity and per cent. of slime are plotted against the tons of feed per day, the moisture remaining constant at 60 per cent. The limits of the curves in each case represent the greatest tonnage that could be forced through the mills and maintained for any length of time. In the case of the peripheral-discharge mill, the tonnage was limited by the fact that when the 10-mesh oversize amounted to more than 25 per cent. of the total, the mill began to fill and finally choked. With the open-end mills, a higher rate of feed was possible, giving a much greater capacity; this was one of

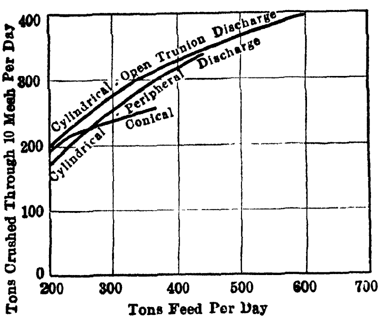


FIG. 7.—EFFECT OF RATE OF FEED ON CAPACITY.

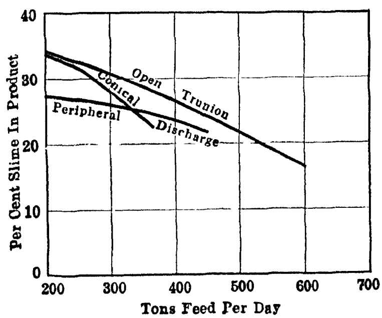


FIG. 8.—EFFECT OF RATE OF FEED ON SLIME.

the main factors that led to the adoption of the open-discharge cylindrical mill. Figs. 7, shows that the maximum capacity of this mill is 390 tons crushed to pass through a 10-mesh screen, against 340 tons for the peripheral-discharge mill, and 255 tons for the conical mill. By passing a very heavy tonnage through the mill, the relative amount of slime is decreased considerably, as shown in Fig. 8; therefore, it is considered best to crowd the mill with a heavy tonnage having a high moisture content and to pass this through the mill rapidly, returning a large amount of oversize. This method of operation also decreases the ball and liner consumption by preventing undue wear of metal on metal.

Table 8 shows a comparison of the three types of ball-mills, with the conditions under which best results were obtained, and the maximum capacity of each mill. The results shown were obtained under closer supervision than is possible in regular operation and which could not normally be maintained. In Table 9 is shown an average condition at a normal operating rate, one that gives a product containing from 20 to 25 per cent. of 10-mesh oversize for each mill, and has been found to be most satisfactory in every way. Under these conditions, the Allis-Chalmers open-discharge mill has the highest capacity, and shows the best ratio of

TABLE 8.—*Operating Conditions and Maximum Capacity of Ball-mills*

	Hardinge Mill	Allis-Chalmers Peripheral-discharge Mill	Allis-Chalmers Open Trunnion Mill
Speed, revolutions per minute	28	23	26
Weight of balls, pounds	10,000	12,000	12,000
Size of balls, inches	4–2½	5–4–2½	5–4–2½
Material of balls	steel	steel	steel
Tons of feed per day	370.0	450.0	600.0
Per cent. moisture	59.3	59.5	60.0
Horsepower consumed	51 6	62 5	69.0
Delays, per cent. time	1.41*	6 44	0 70†
Capacity, tons through 10 mesh.	2 55	3.40	3.90
Tons through 10 mesh per hp.	4.93	5.45	5 65
Per cent. slime in product	22 7	21 88	17 0

* Percentage delay on account of feeders breaking off, 1.35; other causes, 0.06.

† Percentage delay on account of gears, shafts, and countershaft bearings, 0.55; other causes, 0.15.

tons crushed per horsepower. Examination of the products of each mill shows that the Hardinge mill produces more slime lead than the cylindrical type, and that the peripheral-discharge type gives a good product containing a larger amount of granular galena than either of the others. However, mechanical troubles and delays were so great and the capacity so much limited by the grates that it was considered better to use the open trunnion mills, which were installed in all sections of the Bonne Terre mill.

TABLE 9.—*Normal Capacity of Ball-mills*

	Hardinge Mill	Allis-Chalmers Peripheral-discharge Mill	Allis-Chalmers Open Trunnion Mill
Tons of feed per day	300 0	375.0	450 0
Per cent. moisture	60 0	60.0	60.0
Horsepower	51.6	62 5	69.0
Capacity, tons through 10-mesh	237.0	305 0	345 0
Tons through 10-mesh per horsepower	4 60	4.88	5.00
Per cent. slime in product	28.2	24 2	24.1

METALLURGICAL TESTS

In the investigations to determine the exact sphere of ball-mills, as applicable to the flow sheet shown in Fig. 2, the following three schemes were tried: (1) The ball-mill to crush the jig and table middlings for retreatment by tables and flotation, the jigs continuing to make final

concentrates and tailings; (2) the ball-mill to crush jig tailings and table middlings, the jigs acting as roughers, and making final concentrates; (3) the ball-mill to crush oversize of the 2-mm. screens for treatment by tables and flotation, eliminating the jigs entirely.

The first scheme, the flow sheet for which is shown in Fig. 9, simply replaced the middlings rolls by ball-mills. These rolls received about 200 tons of jig middlings and 30 tons of table middlings per section, and al-

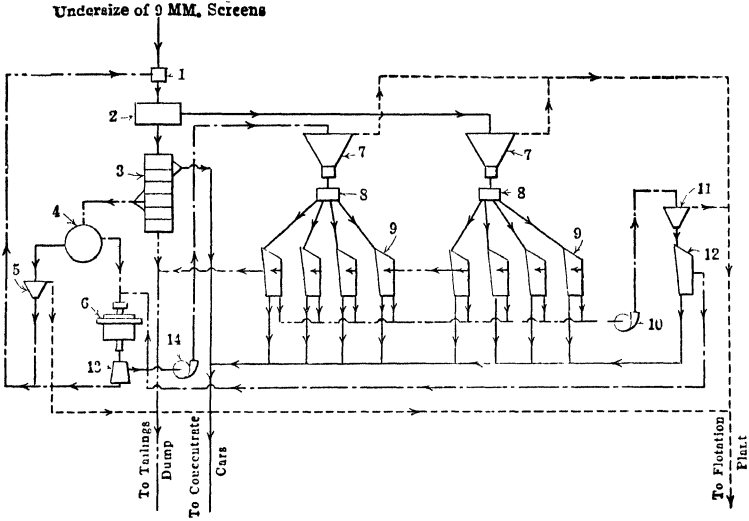


FIG. 9.—OUTLINE OF FLOW SHEET WITH BALL-MILLS CRUSHING MIDDLINGS.

1. Elevator
2. 2-mm. screens
3. Hancock jig
4. Dewatering wheel
5. Settling tank
6. Ball-mill
7. Deslimer
8. Distributor
9. Wilfley-Butchart tables
10. Pump
11. Dewatering box
12. Wilfley-Butchart table
13. 2-mm. screen
14. Pump

though they did fairly good work on the coarse sizes (3 to 6 mesh), their crushing of finer sizes was very unsatisfactory. The result was a large accumulation of fine middlings in the circuit, leading to overloading of all machines, and correspondingly poor work. The introduction of the ball-mill eliminated this accumulation, increasing the capacity of the mill by 25 per cent., and decreasing the jig tailings from 0.85 to 0.75 per cent. lead, and the table tailings from 0.60 to 0.55 per cent. lead. Also, it was found that a tailing assaying 0.30 per cent. could be produced on the tables treating ball-mill product. This was due to the larger amount of material in the finer sizes (see Table 10), and the correspondingly larger percentage of free lead. Details of the metallurgical results are shown in Table 11, showing an average tailing from the total mill of 0.50 per cent. lead, with a recovery of 88.08 per cent.

TABLE 10.—*Screen Analysis of Feed to Tables*

Mesh		Tables Treating Roll Product	Tables Treating Ball-mill Product
On	10	1.8	1.4
	14	12 2	4.6
	20	17.4	8.2
	28	19.9	14.5
	35	13 5	15.1
	48	10.2	16.1
	65	7.2	18.2
	100	6.6	13 1
	150	6.8	5.3
	200	1.5	1.6
Through	200	2.9	1 9
Total..................		100.0	100.0

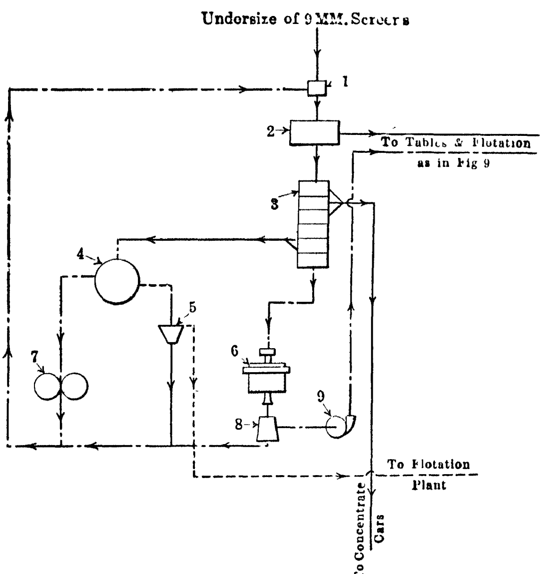


FIG. 10.—OUTLINE OF FLOW SHEET WITH BALL-MILLS CRUSHING JIG TAILINGS.

1. Elevator
2. 2-mm. screens
3. Hancock jig
4. Dewatering wheel
5. Settling tank
6. Ball-mill
7. Rolls
8. 2-mm. screen
9. Pump

The amount of lead going to the flotation plant was considerably increased by this arrangement, and it was thought that a large amount of lead that might be recovered by the jigs was being crushed unnecessarily. To remedy this, the jig was arranged as a rougher, making a final con-

centrate, a high-grade middling which was crushed in the rolls and returned to the original circuit, and a low-grade middling or tailing which was crushed in the ball-mill and retreated by tables and flotation. The flow sheet is shown in Fig. 10, and the metallurgical results in Table 12. The recovery was increased considerably by this arrangement, and the

TABLE 11.—*Results of Operations with Ball-mills Crushing Jig Middlings with Arrangement Shown in Fig.* 9

Material		Tons per 24 Hr.	Assay Per Cent. Lead	Lead Content Tons per 24 Hr.	Distribution of Lead Contents in Final Products Per Cent.
Ore		2,650.00	4.00	106.000	100.00
Concentrates	Jig	8.50	70.00	5.950	5.62
	Table	84 88	70 00	59 415	56.06
	Flotation	46 67	60.00	28 000	26 40
	Total	140.05	66 70	93.365	88 08
Tailings	Jig	740 00	0 75	6.150	5 80
	Table	1,069 95	0 41	4.385	4 14
	Flotation	700.00	0.30	2.100	1 98
	Total	2,509 95	0 50	12 635	11.92

TABLE 12.—*Results of Operations with Ball-mills Crushing Jig Tailings with Arrangement Shown in Fig.* 10

Material		Tons per 24 Hr.	Assay Per Cent. Lead	Lead Content Tons per 24 Hr.	Distribution of Lead Contents in Final Products, Per Cent
Ore		2,650 00	4.00	106.000	100.00
Concentrates	Jig	19.70	70.00	13.780	13.00
	Table	81 00	70.00	56.700	53 45
	Flotation	42.80	60 00	25.680	24.25
	Total	143.50	67.10	96.160	90.70
Tailings	Jig				
	Table	1,744.50	0.43	7.560	7.14
	Flotation	762.00	0.30	2.280	2.16
	Total	2,506.50	0.39	9.840	9.30

amount of slime lead was smaller than by the other scheme. However, the milling cost was very much higher, as will be shown later.

The third scheme, which is shown in Fig. 11, was attractive because of the possibility of a large reduction in the milling cost and a much simplified flow sheet. It would not only eliminate the jigs, but much accessory equipment, such as middling rolls, dewatering wheels, boxes, launders, etc. On the other hand, it promised to give an excessive amount of flotation lead; this point was carefully watched through all of the tests.

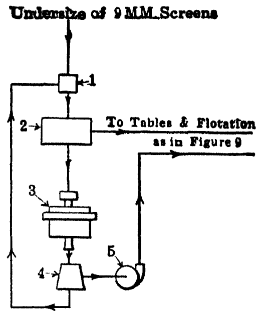


FIG. 11.—OUTLINE OF FLOW SHEET WITHOUT JIGS.

1. Elevator
2. 2-mm. screens
3. Ball-mill
4. 2-mm. screens
5. Pump

Results of preliminary experiments on one mill section were very satisfactory, indicating a smaller amount of lead going to the flotation plant than was expected. An extended test was therefore made on the entire mill without jigs, in order to determine the tonnage that could be handled and the distribution of lead between tables and flotation. Details of the results are shown in Table 13.

Examination of the mill tailings invariably shows that the coarse sizes carry higher lead content than the fine, and it is obvious that, within

TABLE 13.—*Results of Operations Without Jigs*

Material		Tons per 24 Hr.	Assay Per Cent. Lead	Lead Content Tons per 24 Hr.	Distribution of Lead Contents in Final Products, Per Cent.
Ore		2,650.00	4 00	106.00	100.00
Concentrates	Table	95 14	70.00	66.597	62 78
	Flotation	50.10	60.00	30.060	28.40
	Total	145 24	66 50	96 657	91 18
Tailings	Table	1,699 76	0.41	6.928	6.54
	Flotation	805.00	0.30	2 415	2.28
	Total	2,504.76	0.37	9.343	8.82

certain limits, the recovery will increase as crushing is carried down to finer sizes. The jigs will make a tailing assaying 0.75 per cent. lead, the tables treating roll product 0.55 per cent., the tables treating ball-mill product 0.30 per cent., and flotation 0.30 per cent. Obviously, if the crushing were carried so far that all the material was as fine as the combined feed to the ball-mill tables and flotation, the total mill tailing would assay 0.30 per cent., and the recovery would be approximately 93 per cent.

Further crushing than this would gain nothing, unless the grade of the flotation tailing was lower. In the four arrangements discussed here, crushing has been carried down in successive steps, each step giving a higher mill recovery. Figs. 12 and 13 illustrate the distribution of ore and its lead content in the final mill products, and are interesting as showing the progressive steps in finer crushing, with the decreasing amount of jig tailing, and the increasing amount of flotation tailing. Also, Fig. 13

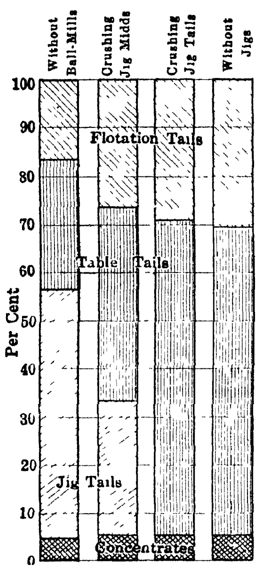


FIG. 12.—DISTRIBUTION OF ORE AS FINAL PRODUCTS.

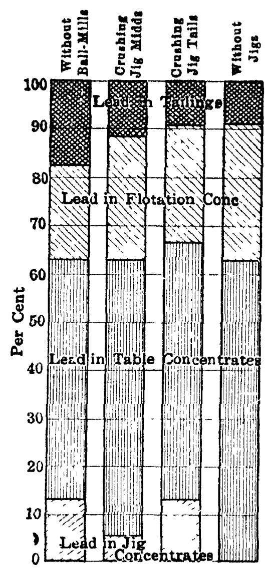


FIG. 13.—DISTRIBUTION OF LEAD IN FINAL PRODUCTS.

shows that the nature of the final concentrates has not been vitally affected. The results obtained without ball-mills, with the ball-mills crushing jig middlings, and without the jigs, show approximately the same amount of coarse concentrates, the increased recovery being in each case obtained by flotation. When the ball-mills are used to crush jig tailings, practically the same recovery is obtained as without the jigs, and the concentrates contain considerably less slime lead. From a purely metallurgical point of view, therefore, the best results are obtained by using the jig as a rougher and crushing jig tailings, but the mill cost is so high as to make this scheme inferior to that of eliminating the jigs.

Economic Considerations

The economic factors relevant to the problem under consideration are cost of milling, and freight, smelter, and royalty charges on concentrates. The last three are obtained by simple calculations for each case, but

determination of mill cost introduces a variety of considerations that must be dealt with in detail. Differences in the cost of power have been obtained by an average of a number of power readings on the motors involved, and an average of the cost of power per kilowatt-hour for the past year. The cost of operating labor has been determined by careful consideration of the labor requirements under each flow sheet, and accurate figures on maintenance have been possible because of the detailed cost-keeping system in use in the mill. This system separates each machine or set of similar machines into operating shop orders, all material and labor for repairs and maintenance on any machine being charged directly to the shop order concerned. In this way accurate maintenance costs are obtained monthly, and it has been possible to obtain, with extreme accuracy, the change in milling cost for any elimination or addition of machinery.

Comparative total costs for each of the schemes under consideration are shown in Table 14. By operating without jigs, the milling cost per

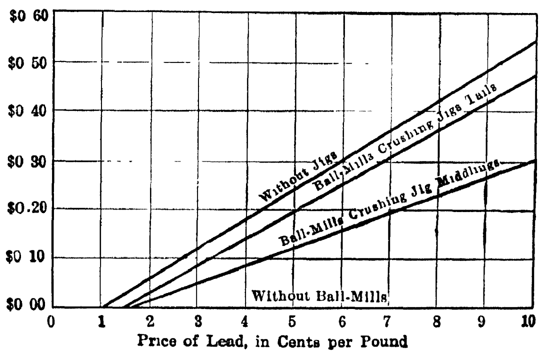


FIG. 14.—DIFFERENTIAL NET VALUE OF PRODUCT PER TON OF ORE AT VARYING PRICE OF LEAD.

ton of ore is at a minimum but other charges are at a maximum. However, the total cost per ton of ore is less than when ball-mills are used to crush jig tailings, and both these arrangements are preferable to the others by virtue of their higher recovery. Final judgment is made on the highest net value of the products, that is, the gross value of recovered lead less milling and other costs; in Fig. 14, this net value is shown under each flow sheet, for all prices of lead. The ball-mills are a decided asset in every case, but operation without jigs is the most economical arrangement. As a result of these investigations and findings, the jigs have been removed from the Bonne Terre mill, which has operated under this arrangement for six months and has given results that have confirmed the experiments, and are highly satisfactory in every way.

TABLE 14.—*Economic Data. Differential Costs per Ton of Ore Milled*

		Without Ball-mills		Ball-mills Crushing Jig Tails		Ball-mills Crushing Jig Middlings		Without Jigs	
		Pounds	Pounds Lead Content	Pounds	Pounds Lead Content	Pounds	Pounds Lead Content	Pounds	Pounds Lead Content
Concentrate produced per ton ore	Jig ..	14.86	10.40	14.87	10.41	6.41	4.49		
	Table ...	57.00	39.90	61.10	42 77	64 10	44.87	71.80	50.26
	Flotation	27.75	16.65	32.30	19.38	35.20	21.12	37.80	22 68
	Total	99 61	66.95	108.27	72 56	105.71	70.48	109 60	72 94
Mill recovery, per cent.....		83.69		90 70		88.08		91.18	
Cost per ton ore difference	Mill.	$0 0067		$0.0409		$0.0144		$0.0000	
	Freight.. . Smelting . Royalty ..	0.0000		0.0485		0.0435		0.0655	
	Total	$0 0067		$0.0894		$0.0579		$0 0655	

Calculation of Ore Tonnage and Grade from Drill-hole Samples

BY JAMES E. HARDING, ANTOFAGASTA, CHILE

(New York Meeting, February, 1921)

THE usual method of sampling mineral deposits is to drill holes and assay the sludge or core. Though the results thus obtained may not represent the true average value of the deposit, it is on these results that estimates of practically all large orebodies are made. In a large number of cases, the results of exploitation fall far short of the estimated value of the orebody.

The standard method of making these estimates is to find the cubic contents of triangular prisms, in the apexes of which the drill holes are placed, by multiplying the surface area by the average depth of the ore in the three drill holes and then multiplying the cubic contents by the specific gravity of the ore or rock to find the tonnage. The assay value is found by dividing the sum of the products of the depth of ore in each hole and the corresponding assay value by the sum of the depths of the ore in the three holes. As the latter part of this method is subject to many mathematical errors, it should not be depended on. Some engineers, therefore, use a discount factor, but as this factor is either arbitrarily selected or is obtained by obscure methods, the results are no better than guesswork.

The standard method is correct when the drill holes are so laid out that the triangular prisms are equilateral in cross-section. But as ore deposits cannot always be drilled into at regular intervals and because of the human factor, if many holes are drilled, triangular prisms of all degrees of angularity are produced. In such cases it is necessary to assume that the influence of the different holes is not the same in all directions, which assumption of course is absurd.

The area of the triangular prism is usually found by scaling; an orebody of sufficient size to justify churn drilling is too large to be handled conveniently on maps having a scale larger than from 1000 to 1 to 1500 to 1 and maps of such scales cannot be accurately measured. A variable is thus introduced in the first operation. In addition, the scaling is frequently done from blueprints, which sometimes shrink as much as 4 per cent. between printing, washing, and drying. The error due to scaling may be eliminated by surveying the location of each hole and

calculating its coördinates; then from these coördinates calculating the length of each line connecting two drill holes.

For this work a rather simple formula has been derived from the regular $a \sin B = b \sin A$ formula.

By subtracting the corresponding coördinates of two holes, the latitude and departure of one from the other is found. As this latitude and departure represent a right triangle and as the sine of 90° is unity, the logarithm of which is 0, the formula $a \sin B = b \sin A$ may be solved and written

$$a = \frac{b}{\sin B}$$

in which a = required distance; b = lesser remainder from subtraction of coördinates; B = angle opposite b.

When the different distances between the various churn drill holes have been found, by the regular formula for the area of a triangle

$$\text{area} = \sqrt{s(s-a)(s-b)(s-c)}$$

the horizontal area of the triangular prisms may easily be calculated.

In actual practice, it will not be necessary to write out any of the factors other than a, b, c, and s, the subtractions and the logarithms. When the area of the prism is thus found, it may be checked fairly closely by scaling a map kept at hand for the purpose, but this check will generally not come out as close as 1 per cent. This method does not take into consideration the fact that, in drilling, the holes may have drifted from the vertical line, thereby increasing or decreasing the actual volume of the prism as depth is attained, but neither does the scaling method.

The unreliability of the standard method of calculation is illustrated by Figs. 1 and 2 which show a trapezoid having in the apexes four holes. Hole *51* had 82.0 m. of ore averaging 2.87 per cent. of copper; hole *137* had 35.0 m. of ore averaging 1.18 per cent. copper; hole *4* had 195.0 m. of ore averaging 2.96 per cent. copper; and hole *134* had 210.0 m. of ore averaging 1.11 per cent. copper. The position of these holes and the corresponding depth of ore and average assay value are given on the figures.

In Fig. 1, the trapezoid is divided into two triangular prisms of scalene cross-section, prism *184* having in its apexes holes *51*, *134* and *137* and prism *183* having in its apexes holes *4*, *134*, and *137*. The area of each prism, which is given in the figure, was calculated by the method just outlined and the results of the calculation of tonnage and grade of the ore content are shown in the accompanying table of calculations. When the trapezoid is divided in this way the two prisms contain 3,099,788.15 metric tons of ore with a metallic content of 55,440.00 tons of copper, or a grade of 1.789 per cent.

In Fig. 2, the same trapezoid is divided in the other possible way: prism *180* having in its apexes holes *4*, *51*, and *137* and prism *181* having in its apexes holes *4*, *51*, and *134*. When thus divided, the table of calculations shows that the two prisms contain 3,144,005.23 metric tons of ore carrying a metallic content of 75,377.53 metric tons of copper and an average grade of 2.398 per cent.

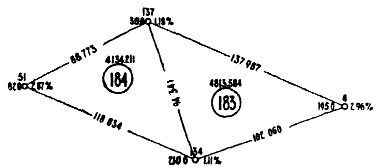


FIG. 1.

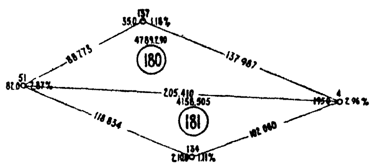


FIG. 2.

Thus, the calculations of amount of copper in the one block of ground, when divided in different ways, instead of agreeing, differ by 19,937.53 metric tons, while the grade differs 0.610 per cent., the second calculations showing 35.63 per cent. more copper than the first. This difference is far more than is permissible, but as far as the calculation goes the one is as correct as the other, if the standard method of calculation is assumed to be correct.

Obviously when two calculations of the same mass, by the same method, show such a large difference and it is known that no mathematical errors were made, one is justified in suspecting that both results may be wrong and that the method is the probable source of error. An effort

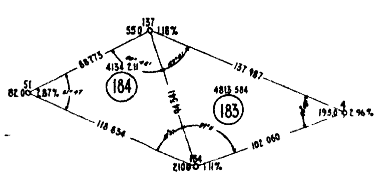


FIG. 3.

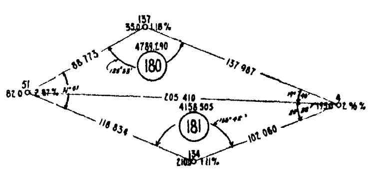


FIG. 4.

has, therefore, been made to devise a method that will give results that will check more closely. From the beginning, it was evident that some factor of correction must be derived.

It is permissible to assume that the importance of a hole in the prism varies as the angle of which it is the apex varies from 60°; that is, if the angle is greater than 60° the importance of the hole should be increased and if the angle is less, the importance of the hole should be diminished; or

$$\frac{x}{\text{Average assay times depth of ore}} = \frac{\text{angle at hole}}{60^\circ}$$

in which x = importance of hole.

Prism No.	Holes No.	Thickness of Ore	Per Cent Cu	Products	Average Per Cent Cu.	Average Thickness of Ore in Meters	Area of Prism in Square Meters	Volume of Prism Cubic Meters	Metric Tons of Ore 2.68 Tons Per Cubic Meter	Metric Tons of Copper
							Fig 1.			
	4	195 0	2 96	577 20						
183	134	210 0	1 11	233 10	1935	146 67	4813 58	706,008 37	1,892,102 43	36,612 18
	137	35 0	1 18	41 30						
		440 0		851 60						
	51	82 0	2 87	235 34						
184	134	210 0	1 11	233 10	1559	109 00	4134 21	450629 00	1,207,685 72	18,827 82
	137	35 0	1 18	41 30						
		327 0		509 74	1789		8947 79	1,156,637 37	3,099,788 15	55,440 00
							Fig 2.			
	4	195 0	2 96	577 20						
180	51	82 0	2 87	235 34	2737	104 00	4789 29	498,086 16	1,334,870 91	36,535 42
	137	35 0	1 18	41 30						
		312 0		853 84						
	4	195 0	2 96	577 20						
181	51	82 0	2 87	235 34	2 147	162 33	4158 50	675,050 12	1,809,134 32	38,842 11
	134	210 0	1 11	233 10						
		487 0		1045 64						
					2398		8947 79	1,173,136 28	3,144,005 23	75,377 53
			Difference		0609			16,498 91	44,217 08	19,937 53
Average of Fig 1 and Fig 2.					2095				3,121,896 69	65408 77
						Or Handled As A Trapezoid				
	51	82 0	2 87	235 34						
	137	35 0	1.18	41 30						
	4	195 0	2 96	577 20	2 082	130 5	8947 79	1,167,687 25	3,129,401 83	65,154 15
	134	210 0	1 11	233 10						
		522 0		1086 94						

Prism No.	Angle	Holes No.	Thickness of Ore	Per Cent Cu	Products	Average Per Cent Cu.	Average Thickness of Ore in Meters	Area of Prism in Square Meters	Volume of Prism Cubic Meters	Metric Tons of Ore 2.68 Tons Per Cubic Meter	Metric Tons of Copper
								Fig 3			
	43°-08'	4	195 0	2 96	414 939						
183	89°-11'	134	210 0	1 11	346 476	1805	146 67	4813 58	706008 37	1892102 43	34152 45
	47°-41'	137	35 0	1 18	32 823						
			440 0		794 238						
	51°-47'	51	82 0	2 87	203 410						
184	47°-31'	134	210 0	1 11	184 604	1355	109 00	4134 21	450629 00	1207685 72	16364 14
	80°-42'	137	35 0	1 18	55 550						
			327 0		443 264	1630		8947 79	1156637 37	3099788 15	50516 59
								Fig 4			
	19°-46'	4	195 0	2 96	190 159						
180	31°-51'	51	82 0	2.87	124 943	1293	104 00	4789 29	498086 16	1334870 91	17259 88
	128°-23'	137	35 0	1 18	88 370						
			312 0		403 472						
	23°-22'	4	195 0	2 96	224 791						
181	19°-56'	51	82 0	2.87	78 184	1713	162 33	4158 50	675050 12	1809134 32	30990 47
	136°-42'	134	210 0	1 11	531 080						
			487 0		834 055						
						1.535		8947 79	1173136 28	3144005 23	48250 35
				Difference		0095			16498 91	44217 08	2266 24
Average of Fig 3 and Fig 4						1582				3121896 69	49383 47
							Or Handled As A Trapezoid				
	51°-47'	51	82 0	2.87	135 409						
	128°-23'	137	35 0	1 18	58 714						
	43°-08'	4	195 0	2 96	276 631	1580	130 5	8947 79	1167687 25	3129401 83	49444 55
	136°-42'	134	210 0	1 11	354 054						
	360°-0'		522 0		825 000						

FIG. 5.—TABLE OF CALCULATIONS.

Solving this proportion gives the applied factor of correction which is substituted in the calculations of grade for the product of the depth times the average assay value as in the standard method of calculation.

As a test, this method is applied to the same trapezoid that was calculated in Figs. 1 and 2; Fig. 3 corresponds to Fig. 1 and Fig. 4 corresponds to Fig. 2. In both cases the prisms are identical, but in Figs. 3 and 4 the angles at the holes are given.

The table of calculations, Fig. 5, shows that in Fig. 3, hole *4* has a product of 414.939 instead of 577.20 as in Fig. 1. This product is the result of solving the proportion

$$\frac{x}{577.80} = \frac{43^\circ 8'}{60^\circ}$$

43° 8′ being the angle at hole *4*. Each of the other holes is treated in the same way and the corrected products are used in determining the average grade of the prism. In Fig. 3, this method gives the content of the trapezoid as 50,515.59 metric tons of copper or an assay value of 1.630 per cent. copper. In Fig. 4, this method gives the trapezoid a content of 48,250.35 metric tons of copper, or an average grade of 1.535 per cent., an assay difference between the two of 0.095 per cent. as against a difference of 0.609 per cent. when the prisms were calculated by the standard method. Now 0.095 per cent. represents 1.92 lb. of copper per ton and 0.609 per cent. represents 12.2 lb. of copper per ton.

As a further check on the method, the trapezoid was calculated as one mass using the standard method as well as the method outlined; these results also are shown in the table of calculations. The closeness of the check to the average in both cases is worthy of note: by the standard method there are three results, no two of which are close enough to be satisfactory, while by the method devised none of the results differs from the others enough to cause any trouble. In this latter case, the angles were taken as proportional to 90° instead of to 60°.

From the way these drill holes, assay values, and prisms were selected, the method advocated produces a result less than the standard method, but a case could be selected, just as easily, in which the results would be be greater. In case hole *4* should fall at the location of hole *134* or *137*, which have angles greater than 90°, the result would be to increase the average assay of the trapezoid. From this, it seems that the standard method is equally liable to err in being too low or too high in different cases.

Probably in calculating an orebody by the standard method the error of the calculated from the actual would not be as great as the off-check between Figs. 1 and 2 on account of compensating errors. Neither in general churn drilling would many prisms develop as scalene as in Fig. 2, but it seems that a method should not be used that can give results that vary as much as the difference between Figs. 1 and 2 when a method is available which checks as closely as do Figs. 3 and 4.

DISCUSSION

WILLIAM YOUNG WESTERVELT, New York, N. Y.—The method referred to as standard is simply one combining mathematical calculations with judgment. Textbooks advise that care be exercised that the triangles do not center unduly around any one hole in an area divided by a series of irregularly placed holes. The author, by applying as factor the amount of spread of the angle of each triangle until he gets around the whole 360 degrees, automatically balances the proper proportion of each triangle to the area surrounding the hole.

In his description of the standard method, the author does not call attention to the fact that it is the custom of engineers to use their judgment in arranging the triangles so as not to lay improper emphasis on a particular hole, as brought out in the author's illustration. In other words, the old method required a reasonable amount of judgment in the division of the area. The new method requires no judgment in making the area divisions. It mathematically applies a proper balancing factor regardless of the arrangement of triangles and is therefore an improvement.

R. S. BOTSFORD, Jamaica, Long Island.—The method proposed is an improvement for the mathematical calculation under the particular conditions where more desirable data are not available. The result of the calculations would be acceptable if the nature of the deposit is disclosed by the holes under consideration, otherwise it would be desirable to have additional holes at doubtful points.

R. M. CATLIN, Franklin Furnace, N. J.—It would seem that in averaging a mass, the more samples taken the better the average is liable to be. Sampling in gold mines is generally somewhat different from the sampling of large masses of ore in base-metal mines. Though the reefs or ledges are small, comparatively, there is usually opportunity to sample numerous cross-sections. On the "Rand" levels were run 100 ft. apart, and average samples taken across the ore every 5 ft. These samples were marked on the assay-plan as a fraction, the numerator representing the pennyweights and the denominator the width in inches. A block was considered developed only when its periphery had been so sampled. In the case of large orebodies and a few samples, a factor of safety would seem advisable. In a multitude of samples there is of course greater accuracy whether they are located at the apexes of triangles, or the center of rectangles.

RALPH E. DAVIS, Platteville, Wis.—The method extensively used in the Joplin and Wisconsin zinc districts was developed because of the extreme irregularity of the orebodies. The drill holes are spotted rather irregularly and adjacent holes are connected by straight lines. Each of

these lines is bisected and a perpendicular line is drawn; a polygon bounded by these perpendicular lines is considered as the area of the ore controlled by one drill hole. The blocks of ore thus formed are calculated on the basis of the one drill hole. In that manner the whole orebody would be calculated. Some of these orebodies would be small, some would be large, but no one drill hole would influence the calculation of any ore excepting the ore immediately surrounding it.

WILFRID S. KIRKE, Potrerillos, Chile (written discussion).—The orebody on which I used this method contains several hundred million tons of ore, and several systems were used to estimate the copper content, viz., the so-called "standard" method of prism calculation; a planimeter method for obtaining areas, checked by logarithms; the Harding angular system of prism calculation, and a very complicated geometrical consecutive method, whereby all the totals of each hole in each prism was carried right through to the last hole. The total copper content, in tons, was then divided by the total tonnage of ore to give the copper per cent. per ton. This latter system is too laborious and tedious for general use, and was used only to check the other methods.

The results showed that the Harding angular system, as modified, checked with the geometrical system to within 1.61 tons of ore in several million tons, and the copper content to 0.00000001 per cent. per ton. The first, or "standard," system, when checked with the geometrical, showed a variation of 138,000 metric tons and a loss in copper content, as calculated, of 0.025 per cent. per ton; the same assay data were used in both cases. The planimeter method for obtaining the total area, in this instance, varied so little from that obtained by coördinates and logarithms as not to cause any trouble at all.

The increase in the tonnage of copper by the Harding angular method proved to be more than 54,000 metric tons, and the percentage of copper per ton was also increased by 0.024 per cent.

I worked out all the prisms, about 300, by the Harding angular method and by the standard method, and checked the results in each case by the geometrical method. My calculations were then checked by others and no variation from the figures given were found. It would seem, therefore, that when one considers the numbers of times in which the calculations automatically check themselves when using the Harding system, that this system as modified, is the most reliable. The angles at the three holes must total 180°. The total products of the three holes must total 1,000, the three areas must check up with the area arrived at either by planimeter or logarithms, and the total volume must agree with the mass total tonnage when multiplied by the specific gravity, etc.

JAMES E. HARDING (author's reply to discussion).—Since this system of prism calculation was published, I have had an opportunity to apply

the system to an asymmetrically drilled orebody, during which some unexpected errors were found. For instance, when a prism apexed on a hole having a zero value, in several instances, the calculation showed ore of a higher tenor than any of the samples from the holes at the corners of the prism; thus, it would seem that there are cases in which the method of calculation advocated is incorrect. To take care of such cases, the following modifications have been devised:

Case 1.—When all holes have positive values, let a, b and c, be the depth of ore in the three holes forming the prism; to find the average depth of the ore influenced by any given hole, use the formula:

$$d = \frac{4a + b + c}{6}$$

Then, as one proceeds from hole to hole, b and c should be substituted, thus:

$$\frac{4b + a + c}{6} \text{ and } \frac{4c + a + b}{6},$$

which will give the average depth of the ore in each hole. These formulas are derived as follows:

$$\frac{a + \frac{a + b}{2} + \frac{a + c}{2}}{3}$$

which, in reality, divides the prism into three prisms.

The area influenced by any given hole is found as follows: (1) Divide angle at hole by 180°. (2) Multiply total area of entire prism by above result. (3) Multiply area found by (2) by the average thickness (found by solution of the formula) of ore in the hole, which thus transforms this smaller prism into volume. (4) Multiply this volume by specific weight of ore, which gives total tonnage of ore in minor prism. (5) Multiply total tonnage of ore in minor prism by copper per cent. in hole, which gives total tonnage of copper in each minor prism.

After treating each hole in the prism in this manner, add the tonnages of ore and the tonnages of copper in each of these minor prisms, which will give the mass tonnage of ore and the mass tonnage of copper in the larger prism. Then, divide the mass tonnage of copper by the mass tonnage of ore, to obtain the correct percentage of copper in the larger prism. The accompanying table shows this complete calculation, which seems to leave no possible room for variation or error.

Case 2.—When one of the holes in a prism has a zero value, the same formula in the form $d = \frac{3a + b}{6}$ is used. For finding the depth of ore influenced by the two holes having values and the percentage of area influenced, the angle at a given hole, containing ore, is divided by the sum of the two angles of the holes in which there is ore.

Specimen Prisms Showing Cases 1, 2, and 3

Prism	Hole	Angle at Hole: Degrees	Angle at Hole: Minutes	Thickness of Ore M.	Per Cent, Copper	Average Thickness M.	Angle/180	Areas, Sq. M.	Volume, Cu. M.	Metric Tons Sp. gr. = 2 68	Metric Tons Copper	Average Per Cent, Copper
						Case 1						
303	158	43	00	124.50	1.73	125.00	0.2389	1,001.82	125,227.50	335,609.70	5,806.05	
	159	30	00	102.00	1.55	113.75	0.1666	698.63	79,469.17	212,977.38	3,301.15	
	160	107	00	150.00	1 44	137.75	0.5945	2,493.02	343,413.50	920,348.18	13,253.01	
		180	00	376.50		376.50	1.0000	4,193.47	548,110.17	1,468,935.26	22,360.21	1.52
						Case 2						
305	159	47	15	102.00	1.55	52.50	0.3430	1,344.60	70,591.50	189,185.22	2,932.37	
	161	90	30	9.00	3.12	21.50	0.6570	2,575.52	55,373.68	148,401.46	4,630.13	
	162	42	15	0.00								
		180	00	111.00		74.00	1.0000	3,920.12	125,965.18	337,586.68	7,562.50	
Two ore holes total		137	45									2 24
						Case 3						
309	116	76	45	0.00								
	141	41	45	0.00								
	142	61	30	190.50	1 34	63.50		4,015.90	255,009.65	683,425.86	9,157.91	
		180	00	190.50	1.34	63.50		4,015.90	255,009.65	683,425.86	9,157.91	1.34

Case 3.—When two holes in the prism have a zero value, the average thickness becomes $\frac{a}{3}$, and the tonnage of ore and tonnage of copper is found by treating the entire prism as a minor prism.

In the discussion, undue emphasis was laid on the matter of "judgment," but judgment that cannot be proved mathematically does not deserve the name of judgment. The matter of apexing prisms on holes already drilled cannot be done at will, and if an orebody is drilled in any other way than at the apexes of 60° triangles, asymmetrical prisms must result, and in plotting such prisms a person must use his judgment. In applying this system to an orebody recently, I found that, in spite of using all the judgment possible, in some cases eight prisms would have to apex at a hole having a higher grade of ore than the average and, in other cases, not more than four prisms were compelled to apex at a hole having a variation from the average grade. As a result, in some cases my calculations were unduly assisted to a higher grade, and in others unduly diminished, which was the reason for this attempt to devise a method that would not be subject to either "judgment" or accident. The results of carrying out this method through a complete orebody of a large tonnage seem to justify the labor involved therein—the tonnage checks with the standard method to within 1.014 per cent.—the standard method showed about 55,000 metric tons less copper and the percentage, by the standard method, was about 0.0242 per cent. less. This is quite a valuable consideration in the case of an orebody of several hundred million tons.

Mining Methods and Costs at the United Verde Mine

By H. DeWitt Smith,* E. M., and W. H. Sirdevan,† E. M., Jerome, Ariz.

(Lake Superior Meeting, August, 1920)

The mine operated by the United Verde Copper Co. is situated near Jerome, Ariz., on the eastern flank of the Black Hills, at an elevation of approximately 5500 ft. (1676 m.) above sea level. The mine and the town of Jerome are located on a steep hillside, which slopes rapidly to the valley of the Verde River. The smelter town of Clarkdale is situated in the Verde valley 4.1 mi. (6.5 km.) distant in an air line from Jerome, and at an elevation of 3560 ft. The two towns are connected by the standard-gage railroad of the Verde Tunnel & Smelter R. R., a subsidiary of the United Verde Copper Co. This road is 11 mi. long and has a compensated 4 per cent. grade. At Clarkdale, it connects with the Verde Valley branch of the Santa Fe R. R. The narrow-gage (36-in.) line of the United Verde & Pacific Ry. was, until 1919, Jerome's only rail connection with a main trunk line. This road connects with a branch line of the Santa Fe system at Jerome Junction, after traversing 26 mi. of mountainous country.

From the time of its location, in 1876, until its purchase by Senator W. A. Clark, of Montana, in 1889, the United Verde mine was worked on a small scale and shipments of high-grade gold-silver ore were made from the surface workings. Until 1894, when the narrow-gage railroad to Jerome Junction was completed, all supplies for the mine and smelter

Table 1.—*Production of United Verde Copper Company*–1900 to 1918

Year	Dry Ore Tons	Fine Copper Pounds	Silver Ounces	Gold Ounces
1900–1913	3,800,232	464,394,541	7,242,383	268,152
1914	391,027	32,545,924	683,417	21,963
1915	491,146	45,127,832	902,881	28,221
1916	694,053	58,299,573	1,030,851	26,416
1917	813,176	71,726,634	1,223,310	29,230
1918	861,250	77,501,595	1,292,109	29,281
Total	7,050,884	749,596,099	12,374,951	403,263

* Superintendent of Mines, United Verde Copper Co.
† Chief Mine Engineer, United Verde Copper Co. Died Feb. 15, 1920.

were hauled by wagon from Granite, a distance of 28 mi., at an average cost of $9 per ton. From 1894, copper production increased steadily, the production for the year 1918 being 77,501,595 pounds.

Table 1 shows the production of the United Verde mine for the period 1900-1918 inclusive. There are no accurate data available for the yearly productions prior to 1900. The approximate total amount of ore mined up to the end of 1918 is 8,200,000 tons. The production of the 5 years, from 1914 to 1918 inclusive, therefore, represents 40 per cent. of the total ore production from the mine. The recovery per ton of ore, for the years 1900, 1910, and 1918, is given in Table 2.

TABLE 2.—*Recovery per Ton of Ore*

Year	Fine Copper Pounds	Silver Ounces	Gold Ounces
1900	162.9	2.10	0 065
1910	106 0	1.55	0 058
1918	91.0	1.50	0.034
Average, 1900 to 1918	106.3	1.75	0.057

GEOLOGY AND ORE OCCURRENCE

The geology of the Jerome district has recently been described in detail by L. E. Reber, Jr.[1] The oldest rock exposed in the district is a greenstone complex, consisting of metamorphosed volcanic flows and agglomerates. Overlying the greenstone is a series of clearly bedded sedimentaries. A period of deformation, which squeezed the bedded sediments into folds trending approximately north 20° West, followed the deposition of these sedimentaries. This deformation was followed by an intrusion of quartz porphyry, presumably a marginal phase of the Bradshaw granite batholith which underlies a large area south of Jerome. Subsequent deformation rendered portions of this porphyry schistose, particularly in the vicinity of the United Verde mine. An augite diorite, which shows no evidence of deformation, intrudes both the quartz porphyry and the older formations. The latest pre-Cambrian formation known in the district is a series of narrow diorite dikes, which cut all the formations noted above. The pre-Cambrian rocks are overlain by a great thickness of Paleozoic sediments, which form a prominent feature of the plateau region of northern Arizona. The outpouring of basaltic lavas of Tertiary age ends the periods of rock formation in the Jerome district.

The most striking structural feature of the district is the great Main or Verde fault, which strikes approximately north 37° west, and has a

[1] Geology and Ore Deposits of Jerome District. This volume, p. 3.

vertical displacement of 1700 ft. (518 m.). The United Verde orebody is located in an exposure of pre-Cambrian rocks west of this fault. The United Verde Extension orebody lies to the east of the fault, covered by 800 ft. of Paleozoic sedimentaries and Tertiary volcanics.

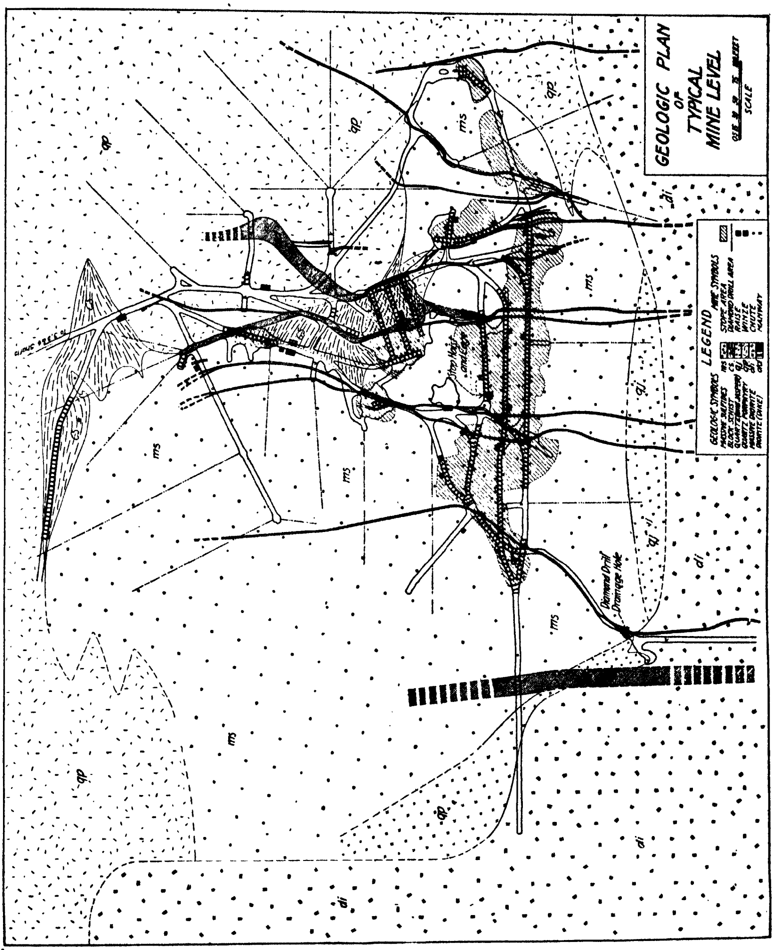


Fig. 1.—Geologic Plan of Typical Mine Level.

The ore deposits of the Jerome district were undoubtedly formed after the intrusion of the augite diorite, and prior to the intrusion of the series of narrow diorite dikes. Consequently they are of pre-Cambrian age, deposited from solutions following in the wake of the Bradshaw batholith.

The ore deposits are in the form of lenses, and of the characteristic schist-replacement type.

The structural conditions at the United Verde mine are shown by the geologic plan of a typical mine level, Fig. 1. The concave margin of the massive augite diorite has formed an impervious trough along which ascending solutions have been localized. The black sedimentary schists within this trough have been irregularly replaced by massive sulfides, in which pyrite predominates, and by jaspery quartz. A later period of chalcopyrite mineralization has somewhat enriched the original sulfide masses and has developed many orebodies in the black-schist areas. The orebodies are steep-dipping lenses, varying in section from a few feet in width and length up to 200 ft. in width by 500 ft. in length (60 by 152 m.). Secondary enrichment is of little importance in the orebodies exposed at the United Verde mine.

Classes of Ore

For the convenience of smelter operation, the ores mined are divided into four classes. Each class must be handled separately in all mining operations.

Oxide Ore.—Oxide ore is mined from the oxidized zone above the 160-ft. (48-m.) level. It contains a small amount of unaltered primary sulfides but, in general, has a low copper content. An average assay shows the precious metal content to be about 0.2 oz. gold and 8.0 oz. silver per ton.

Iron Ore.—Iron ore is mined from the stopes within the massive sulfide areas of the mine. If the silica content is 15 per cent. or more, it is classed as silica ore, and is diverted to the silica-ore bins.

Silica Ore.—The ore from all black-schist stopes and from quartz-porphyry stopes assaying less than 50 per cent. free silica is also classed as silica ore. The analysis given in Table 3 shows that the term silica ore is somewhat of a misnomer, as the iron content is often in excess of the free silica contained.

Converter Ore.—An ore high in free silica is essential for use as a flux in the converters and for fettling the reverberatory furnaces. A small tonnage of ore meeting this requirement is mined from stopes in quartz porphyry near contacts with the sedimentary schists. The main supply, however, is mined from a low-grade, secondary-enrichment orebody in quartz-porphyry, located some distance from the main orebodies, and extending from the 160-ft. to the 600-ft. level.

In addition to these classes of ore, a small amount of precipitate is recovered from the copper-bearing mine waters and shipped directly to the smelter. The average analysis and the tonnage of each class of ore shipped during 1918 are given in Table 3.

TABLE 3.—*Tonnage and Analyses of Ores*, 1918

Class of Ore	Dry Tons	Per Cent. of Total Shipments	Copper, Per Cent.	Gold, Ounces	Silver, Ounces	Iron, Per Cent.	Insoluble, Per Cent.	SiO_2 Per Cent.	Al_2O_3 Per Cent.	Sulfur, Per Cent.	Zinc, Per Cent.
Oxide	22,269	2.59	1.42	0 225	8.37	31.5	40.4	34.1	5.7	4.2	0 2
Iron	422,928	49.10	6.14	0.025	1.94	32 5	11.5	8 8	3.3	37.3	2 6
Silica	324,249	37 65	5.05	0.025	2.06	25 5	24 8	19 3	10.4	21.1	1.2
Converter	91,479	10.62	2 06	0 061	2.32	11 2	71 3	67.0	9.9	3 8	0 3
Precipitates	325	0 04	67 45			4.3	1 8	1.3	2 9	1.0	0 6
Total	861,250	100 00	5.20	0 034	2.19	27 6	23 6	19 6	6 75	26 8	1.77

MINERALOGICAL COMPOSITION OF ORES

From the analyses given in Table 3, the approximate mineralogical composition of each class of ore, expressed in per cent., is given by L. E. Reber, Jr., as shown in Table 4.

TABLE 4.—*Mineralogical Composition of Ores*

	Ores			
	Oxide, Per Cent.	Iron, Per Cent	Silica, Per Cent.	Converter, Per Cent.
Pyrite	5	59	28	5 75
Chalcopyrite		18	16	1 00
Sphalerite		4	2	0 50
Chalcocite	2			1 50
Cuprite, native copper and carbonates				0 25
Cuprite	1			
Sulfates and carbonates of iron, copper, and lime	3			
Iron oxides	46	2	2	11 00
Ferruginous chlorite			38	6 00
Quartz	28	5	5	54.00
Silicates (sericite, hornblende, kaolin, etc.)	15	10	9	20 00
Calcite and siderite		2		
Total, per cent.	100	100	100	100.00

STOPING AREAS

The total area of stoping ground developed to date on the sill floors of four representative lower levels is given in Table 5. These areas do not include material averaging less than 2.5 per cent. copper. To maintain the ore reserves at their present figure and to produce 80,000,000

lb. of copper per year, it will be necessary to average about 100 ft. of development in depth each year.

TABLE 5.—*Stoping Areas, Four Levels*

Stoping Area in Square Feet	Average Grade Per Cent. Copper	Tons of Ore per Foot of Depth	Tons of Copper per Foot of Depth
37,520	5.60	4,461	249.8
36,090	5 18	3,902	202 1
48,860	7.99	6,138	490 4
88,650	8.57	10,811	926 5
Average....49,345	7.06	5,881	415.2

UNIT COSTS OF LABOR AND SUPPLIES

The mining costs given are representative for work performed during the year 1918. General conditions during 1918, particularly in regard to labor, were unfavorable for either low costs or high efficiency. However, the figures will show the relative costs of the various detailed mine operations The average wage scale for the year 1918, covering the major occupations, and the wage scale in effect December, 1919, are shown in Table 6.

TABLE 6.—*Jerome Wage Scale*

Occupation	Average Year 1918	Dec 1919	Occupation	Average Year 1918	Dec 1919
Miners . .	$5 46	$6 10	Average wages of all mine employees	$5.61	$6 01
Carmen and muckers	5 46	5 60			
Timbermen	5 96	6 35	Employees on mine payroll	976	810
Motormen .	5 96	6.35			
Timber helpers .	5 46	5.85	Dry tons of ore per man-shift...........	2 42	2.46
Switchmen	5.46	5 85			
Surface roustabouts.	4 21	4 20			

The average costs of mining supplies for the year 1918 delivered at Jerome, as compared with present (December, 1919) costs, are shown in Table 7.

ACKNOWLEDGMENT

In the following description of mining methods and costs at the United Verde mine, details of methods already fully described in the technical

TABLE 7.—*Cost of Supplies*

	Cost Delivered at Jerome	
	Average, 1918	December, 1919
Explosives:		
35 per cent. ammonia, per pound	$0.20	$0 18
50 per cent. ammonia, per pound	0 23	0.19
35 per cent. gelatin, per pound	0 22	0 19
50 per cent. gelatin, per pound	0 27	0 21
8-X caps, per 100	2.17	2 40
Dreadnaught fuse, per 1000 ft.	6 40	7 17
Timber:		
Lagging, native pine, per M (or 1000 bd. ft.)	17.33	27.30
Cribbing, native pine, per M.	20 40	36 65
Framed round timber, native pine, per linear foot.	0.174	0 186
Miscellaneous lumber, native pine per M.	21.15	32.65
Miscellaneous lumber, Oregon pine per M	36.40	45 75
Drill steel:		
1¼ in. hollow round, per pound	0 19	0 19
⅞ in. hollow quarter octagon per pound	0.22	0 236
Carbide, per hundredweight	5 50	5.83
Fuel oil, per barrel	2 42	3 00
Electric power, per kilowatt-hour	0 0182	0 0135
Rails, mine, per hundredweight	4 25	3 75
Pipe, 2-in., per foot	0 235	0 34

literature have been omitted, except where required for clearness of text. Detailed descriptions of mechanical devices and of the surface plant are likewise not within the scope of this article.

The authors wish to acknowledge their indebtedness to the operating and engineering staffs of the United Verde Copper Co. for assistance in the compilation of the data presented in this paper; particularly to, Robert E. Tally, assistant general manager, and to H. W. Seamon, efficiency engineer.

Exploration and Development

All present exploration and development work, stoping methods, tramming, hoisting, ventilation system, etc. are planned with the definite purpose of providing and maintaining an underground production of 3000 tons of ore per day, with the possibility of an increase to 4500 tons per day, if mine and market conditions should warrant this output.

In order to maintain an underground production of 3000 tons of ore per day, it is necessary to develop 100 ft. (30 m.) in depth every year, and to drive 25,000 ft. of drifts and raises for exploration and develop-

ment work. During the last few years, an average of 34 tons of ore has been developed per foot of development work. This high figure has been due to the character of the orebodies, which are large, persistent, and concentrated within a relatively small area. This advantage is partly offset by a high average cost for all exploration and development work.

Shafts and Shaft Equipment

At the present time there are three working shafts at the United Verde mine. These shafts are called No. 3, No. 4, and No. 5. A fourth

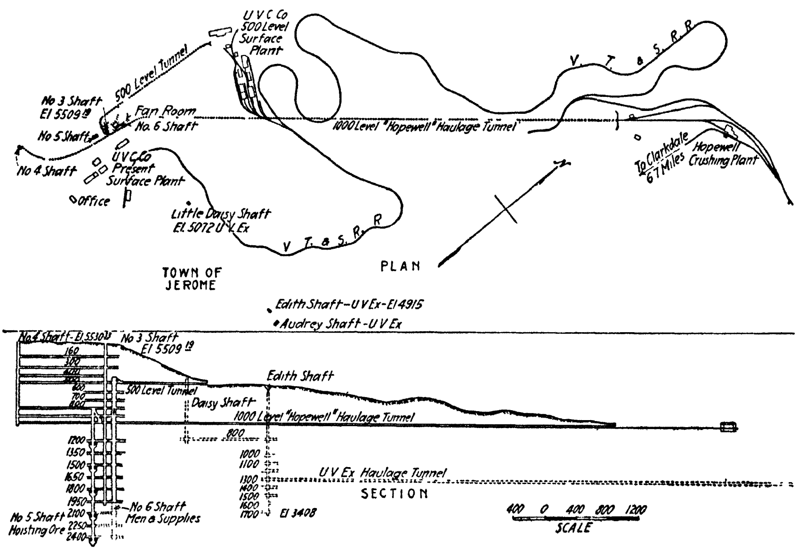


FIG. 2.—LOCATIONS OF SHAFTS AND TUNNELS.

shaft, No. 6, is being concreted and should be in operation by December, 1920. The relative locations of all shafts are shown in plan and section in Fig. 2.

No. 3 *Shaft.*—No. 3 shaft, 5 ft. by 15 ft. 2 in. (1.5 by 4.6 m.) in the clear, is timbered, and has three compartments. It extends from the surface to the 1950-ft. level (594 m.). At this shaft two double-decked cages are operated in balance by a 20 in. by 48 in. (50.8 by 121.9 cm.) duplex, direct-acting, double-reel, Corliss steam hoist of Webster, Camp & Lane manufacture. An unbalanced "chaser" cage is operated in the third compartment by a 16 in. by 30 in. duplex, direct-acting, single-reel hoist of the same manufacture. This "chaser" cage is used only

when the shifts are being lowered or hoisted, and during periods of shaft repair work.

No. 4 Shaft.—No. 4 shaft is timbered with 10 in. by 10 in. Oregon pine, and has two compartments, each 4 ft. 4 in. by 5 ft. (1.3 by 1.5 m.) in the clear. It extends from the surface to the 1000-ft. level (304 m.). Two double-deck cages with platforms 3 ft. 4 in. by 4 ft. 7 in. (1 by 1.38 m.) are operated in balance by an Allis 20 in. by 48 in. (50.8 by 121.9 cm.) duplex, direct-acting, double-reel steam hoist. At the present time this shaft is chiefly used for lowering oxide ore to the 900-ft. level for transfer to standard-gage cars in the Hopewell Tunnel. This ore cannot be handled through the regular transfer raises on account of its sticky nature.

Disadvantages in Operation of Shafts No. 3 *and No.* 4.—Prior to September, 1918, all men, timber, and supplies were handled through shafts Nos. 3 and 4. No. 3 shaft also hoisted all ore mined below the 1000-ft. level to the 900-ft. level for transfer to the Hopewell tunnel loading bins. For the first nine months of 1918, the average tonnage hoisted per day was 1430 tons of ore and 40 tons of waste. The dimensions of the cage platform made it necessary to use cars of 18 cu. ft. (0.5 cu. m.) capacity throughout the mine. The average time required to lower or to hoist a shift of 380 men was 35 min. The entire graveyard shift was required for the handling of timber, drill steel, powder, and miscellaneous supplies. Consequently, the ore had to be hoisted at top speed in order to handle the required tonnage in the time left for this work.

A fire in No. 3 shaft meant the cessation of all ore production from the lower levels. The location of this shaft was also undesirable for a permanent working shaft because the upper part was in moving ground, which made it difficult to keep the timbers in alignment, and about 95,000 tons of 6-per cent. ore had to be left unmined on account of its proximity to this shaft. Enlargement and concreting, as had been done at the Sacramento shaft at Bisbee in somewhat similar circumstances, therefore was not feasible. To overcome these difficulties, No. 5 shaft was designed to provide increased ore-hoisting capacity, and No. 6 shaft was planned to handle all men, timber and supplies.

No. 5 *Shaft for Handling Ore.*—No. 5 shaft is 5 ft. by 16 ft. (1.5 by 4.8 m.) in the clear, and has three compartments. It extends from the 800-ft. (243 m.) level to the 2500-ft. level and has a concrete lining that has a minimum thickness of 12 in. The hoisting equipment has been described in detail by T. D. Hawkins.[2] It is designed to operate two skips in balance to a depth of 2200 ft. below the 1000-ft. level. The maximum rope speed is 1000 ft. per min., and the maximum capacity 5500 tons per 24 hr. In emergencies, the hoist will operate unbalanced

[2] The United Verde Underground Hoist. *Eng. & Min. Jnl.* (Apr. 7, 1917) **103,** 598–601.

for a period of 4 hr. Full speed is attained in 10 sec. and retardation to rest in 5 sec.

The hoist is an Allis-Chalmers, double-drum, single-reduction geared type, driven through a flexible coupling by a 650-hp. direct-current motor, with current at 500 volts, and a normal speed of 300 r.p.m. The drums are cylindrical, 10 ft. in diameter with a 5-ft. face (3 by 1.5 m.) grooved for 2500 ft. (762 m.) of 1⅜ in. (35 mm.) extra plow-steel, 6 by 19, hoisting rope, in two layers. Each drum has a double-disk, multiple-arm, friction clutch and a parallel motion post brake, which are operated by auxiliary engines actuated by oil under pressure. Power is supplied to the hoist motor by a 695 r.p.m. flywheel motor-generator set, consisting of one 700-hp. wound-rotor induction motor, one 600-kw., 500-volt, direct-current generator, one 10-kw., 250-volt exciter, and a steel plate flywheel, 9½ ft. (2.85 m.) in diameter, weighing approximately 40,000 lb. (18,143 kg.). The starter and slip regulator are of the standard liquid type, designed to start the set from rest. This regulator varies the resistance in the secondary circuit of the motor as the service demands, permitting the flywheel to deliver or store energy as required.

In addition to an overwinding device on the hoist indicator, two limit switches located in the head frame are operated by the up-coming skip and automatically stop the hoist if the overwinding device has failed to function. An emergency switch, which cuts off the current and applies the brakes, is located near the operator. In case men are to be hoisted, the speed is limited to 800 ft. per min. by the insertion of a speed-limiting resistance by the operator.

No. 5 hoist room is located on the 1000-ft. level. It is 47 ft. by 81 ft. in section by 22 ft. in height (14.3 by 24.6 by 6.7 m.) and is lined with reinforced concrete. A 20-ton crane facilitates the rapid handling of hoist-room equipment for repairs. The self-dumping skips used in No. 5 shaft have a capacity of 112 cu. ft. (3.1 cu. m.) equivalent to 8.0 tons of "iron" ore, 6.5 tons of "silica" ore, and 5.8 tons of waste. They are of rugged construction and weigh, approximately, 12,000 lb. (5443 kg.). Fig 3 shows a plan and elevation of one of these skips.

The level interval below the 1200-ft. (365 m.) level is 150 ft. (45 m.) and motor haulage is provided on every other level. The skip pockets are therefore located only on the 1200-ft., 1500-ft., and 1800-ft. levels, with pockets in course of construction on the 2100-ft. and 2400-ft. levels. Two pockets, each with a capacity of 300 to 500 tons, are provided at No. 5 shaft on each of these levels. These pockets are ordinarily used for handling the iron and silica ores. Waste from development work on operating levels is transferred through raises to the stopes below to be used as filling. Waste from development work below the 1950-ft. level is stored in one of the temporary shaft pockets on that level, the other pocket being used for ore. The relatively small amount of converter

ore mined below the 1000-ft. level is stored in raises until it is convenient to empty one of the skip pockets for this service. In this way no difficulty is encountered in handling three classes of ore and the waste from development work through the two skip pockets on each haulage level.

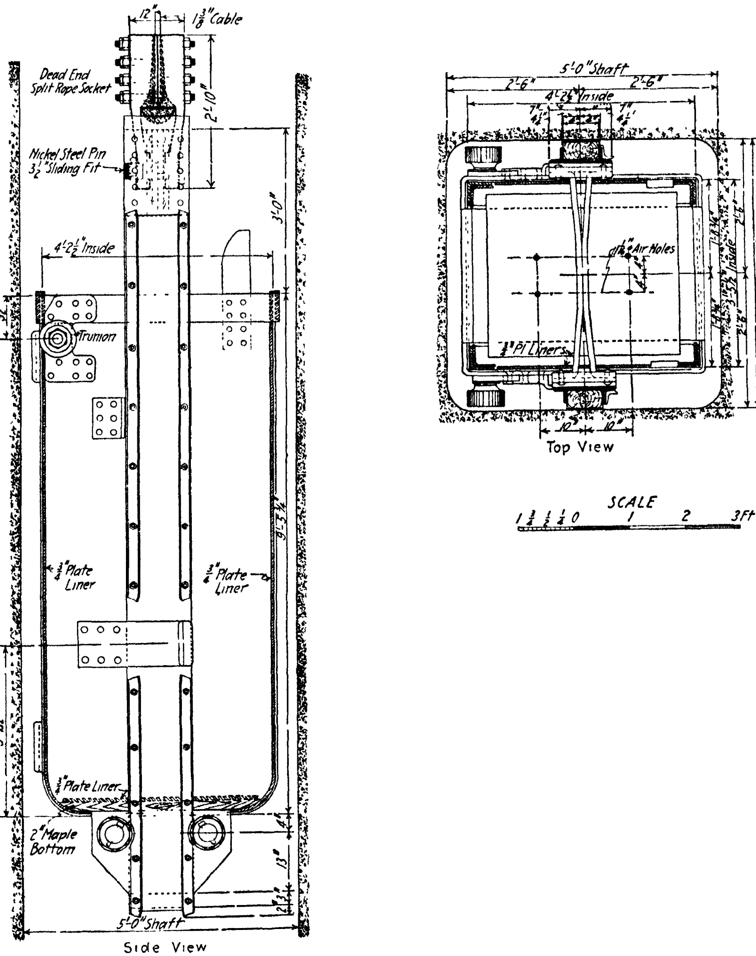


FIG. 3.—SELF-DUMPING SKIP—CAPACITY 112 CUBIC FEET.

The arrangement of the loading pockets follows the general practice in the camps of southern Arizona. Fig. 4 shows a section and a front view of one of these pockets. The ore passes from the loading pocket, through an air-operated, undercut, arc gate to a cartridge of 112 cu. ft. (3.1 cu. m.) capacity, which is the desired skipload. An interlocking

valve prevents the opening of the lower, or cartridge, gate until the upper, or pocket, gate is completely closed.

From the No. 5 hoist room on the 1000-ft. level, the cableway extends at an angle of 60° to the head frame at the 800-ft. level. The hoisting ropes are supported on idlers spaced 34 ft. (10.3 m.) center to center. The sheaves are 10 ft. in diameter and operate in bronze journals.

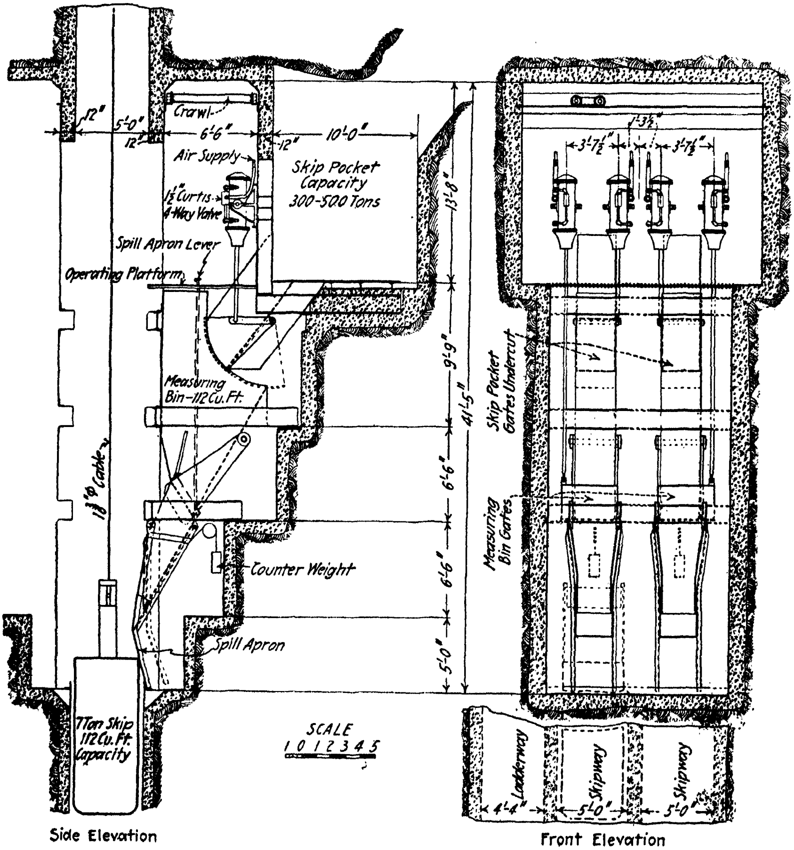


FIG. 4.—LOADING POCKET—NO. 5 SHAFT.

The skip dumps into a bin designed to absorb the shock of the falling ore. The ore then passes through an electrically driven distributor located on the 860-ft. level and controlled by the hoist operator from his platform. This distributor travels over an arc of about 60° and can discharge into any one of three chutes connecting with the 1000-ft. level loading bins. A series of colored pilot lights on the switchboard in

the hoist room indicates the position of the distributor. At the bottom of one of these chutes, there is a bypass arrangement by means of which all waste hoisted through No. 5 shaft is diverted to the 1200-ft. level for use as stope filling.

No. 6 Shaft, for Handling Men and Supplies.—No. 6 shaft is 13 ft. by 13 ft. (3.9 by 3.9 m.) in the clear and has three compartments. The cage compartment is 8 ft. by 13 ft., the pipe compartment 4 ft. by 9 ft. 4 in., and the counterbalance compartment 4 ft. by 3 ft. The shaft extends from the 400-ft. level to the 1950-ft. level, and is to be continued to the 2400-ft. level. Concreting to the 1950-ft. level should be completed by October, 1920. The concrete lining has a minimum thickness of 8 in. (20.3 cm.).

The hoisting equipment is designed to operate a double-deck cage, in balance with a counterweight, at a rope speed of 800 ft. per min. The acceleration is 6 sec., and retardation 5 sec. The hoist is of Nordberg manufacture and has a single cylindrical drum 12 ft. (3.6 m.) diameter with a 6-ft. face. The drum is spirally grooved for 2500 ft. of 1¾ in. (1.9 cm.) extra plow-steel rope in two layers. A reel for ½ in. by 5½ in. flat rope for the counterweight is rigidly coupled to the main drum. The brakes are of the parallel-motion, gravity, post type, set by a deadweight and released by a hydraulic thrust cylinder. The hoist is driven through a flexible coupling by a 350-hp. wound-rotor, induction motor at a normal speed of 360 r.p.m.

No. 6 hoist room is located on the 500-ft. level. It is 44 ft. by 45 ft. in section and 26 ft. in height (13.4 by 13.7 by 7.9 m.) and is lined with reinforced concrete. The cableway extends at an angle of 32° from the hoist room to the head frame at the 400-ft. level. The sheaves for the 1¾ in. round rope and for the ½ by 5½ in. flat rope are 10 ft. and 8 ft. in diameter, respectively. The shaft station at the 500-ft. level is so constructed that both cage decks may be loaded or discharged simultaneously. These cages have a capacity for 96 men. They also permit the transfer of steel and mine timbers without rehandling at the shaft collar and at the level stations below.

General Hoisting Costs

The average cost of hoisting ore and waste through No. 5 shaft in the months of August, September, and October, 1919, was as follows, the total tonnage hoisted being 110,487 tons:

	Cost Per Ton Hoisted
Labor	$0 021
Supplies	0.001
Electric power	0.009
Repairs	0.003
Total	$0.034

The total cost of all hoisting for the year 1918, when 530,470 dry tons were hoisted, is given in Table 8:

TABLE 8.—*Total Hoisting Costs,* 1918

	Total Cost Per Ton of Ore Hoisted
Labor	$0.181
Supplies	0.004
Electric power	0.003
Boiler plant	0.205
Repairs	0.037
Total cost per ton of ore hoisted	$0.430
Total cost per ton of ore shipped to smelter	$0.265

These costs clearly indicate the uneconomical conditions of hoisting, which should be eliminated with the operation of No. 5 and No. 6 shafts.

Shaft Raising and Sinking

Raising and Sinking No. 5 *Shaft.*—No. 5 shaft was raised from the 1950-ft. to the 860-ft. level, a total distance of 1090 ft. The rock section excavated measured 7 ft. by 17 ft. (2 by 5 m.)

This shaft was then sunk from the 1950-ft. level to a depth of 2500 ft., a total distance of 550 ft. The rock section excavated by sinking measured 7 ft. by 18 ft. The shaft was first securely bulkheaded above the 1950-ft. level and then was sunk full size for a distance of 20 ft. Two small skip pockets were excavated with approximately 70 tons capacity, so that all shaft waste could be handled directly through No. 5 shaft. The shaft was then sunk for 30 ft. the size of the manway only; widened to the full cross-section below this depth and the timber bulkhead above the 1950-ft. level then removed. The usual rock pentice, in this case 30 ft. in thickness, was left for the protection of the shaft men in further sinking. A 50-hp. electric hoist, with a capacity of 3500 lb. (1587 kg.) at a rope speed of 300 ft. (91 m.) per min., was installed opposite the manway compartment. A 10 cu. ft. (0.28 cu. m.) bucket attached to a safety crosshead with bonnet was used in sinking. All mining, mucking, and timbering was done on two shifts with crews of four men. The rock encountered was tough quartz porphyry. A V cut, drilled with Jackhamers, using thirty-eight 5-ft. holes, gave an average advance of 4 ft. per round. Delay-action detonators for blasting were used with good results. Shaft sets were of 8 in. by 8 in. Oregon pine, spaced 6 ft. from center to center. The small amount of water encountered in sinking was hoisted in the bucket at the beginning of each shift. Comparative costs and data for the raising and sinking of No. 5 shaft are shown in Table 9.

TABLE 9.—*Costs and Data of Raising and Sinking No. 5 Shaft*

	Raising		Sinking	
	Cost per Foot	Cost Per Cubic Yard	Cost per Foot	Cost Per Cubic Yard
Labor	$12.91		$38.41*	
Explosives	3.08		4.43	
Timber	2.60		5.39	
Compressed air	4.20		5 83	
Supplies and repairs	6.20		6.10	
Total cost	$28.99	$6 58	$60.16	$12.89
Total advance, feet	1090		510	
Section excavated, feet	7 × 17		7 × 18	
Drills used	Ingersoll-CCW 11		Sullivan DP 33 Ingersoll DCRW 430	
Average depth of hole, feet	5 5		5 0	
Average number of holes per round	34		38	
Distance drilled per round	187		190	
Advance per round, feet	4 25		4 0	
Distance drilled per foot advance, feet	44 0		47 5	

* Labor costs in sinking include charges for hoistman and topman at 1950-ft level.

Raising No. 6 Shaft.—No. 6 shaft was raised from the 1500-ft. level in two lifts; one from the 1500-ft. to the 900-ft. level, and the other from the 900-ft. to the 400-ft. level. On each lift a cribbed chute and manway, 5 by 10 ft. (1.5 by 3 m.) was carried up along one wall. The second com-

TABLE 10.—*Cost and Data of Raising No. 6 Shaft*

	Cost per Foot	Cost per Cu Yd.
Labor	$23 88	
Explosives	5 14	
Timber	3.85	
Compressed air	2 62	
Drill repairs	4 87	
Supplies	0 41	
Total cost	$40 77	$5.34

Total advance, feet	1070
Section excavated, minimum dimensions	14 ft. 4 in. by 14 ft. 4 in.
Drills used	Sullivan BT 44 and Ingersoll CCW 11
Average depth of hole, feet	5.5
Average number of holes per round	40
Distance drilled per round, feet	220
Advance per round, feet	4 75
Distance drilled per foot of advance, feet	46.3

partment gave the advantage of better ventilation and a much longer base for the maintenance of survey lines than was possible with a single cribbed manway. The rock section, 14 ft. 4 in. by 14 ft. 4 in. (4.3 by 4.3 m.) as a minimum, was excavated by contractors with an average overbreak of 2¾ in. (7 cm.) in the tough quartz porphyry of the lower lift and 4¾ in. (12 cm.) in the schist and fractured diorite of the upper lift. This small overbreak was obtained by careful placing of all holes by the contractors, to avoid the heavy penalty for any overbreak exceeding 6 in. The costs of raising No. 6 shaft and drilling data for the work are shown in Table 10. The costs for drill repairs is high because this shaft work was done during a shutdown in the spring of 1919, when a large overhead expense was charged out on the basis of drill shifts worked.

Level Interval

The most economical interval between levels has been found to be 150 ft. (45 m.). A closer spacing increases proportionately the cost of level development per ton of ore mined. In mining under a filled stope, it is necessary to extract three or four floors directly beneath the level by the use of square sets, a more expensive mining method than the horizontal cut-and-fill method generally used for the remainder of the stope. With a level interval of 100 ft., approximately 20 per cent. of the total ore must be mined by the square-set method, while the corresponding ratio is only 15 per cent. when the level interval is 150 feet.

A 200-ft. level interval (1000 ft. to 1200 ft.) was found unsatisfactory. The increased cost for raise development, the high cost for maintenance of the ore chutes in the stopes, and the necessity of providing a sublevel for development halfway between the two levels, more than offset any advantage gained by using the longer lift.

Drifts, Crosscuts, and Gangways

Size and Timbering.—As cars of the same size are used for hand tramming and for motor haulage, all drifts and crosscuts have the same cross-section, 6 ft. in width by 8 ft. (1.8 by 2.4 m.) in height in the rock section, and 5 ft. by 7 ft. (1.5 by 2.1 m.) in the clear where timbered. All drifts and crosscuts are given a grade of 0.4 per cent. in favor of the load.

Gangways under stopes are first driven 7 ft. wide by 8 ft. high, and a second cut 7 ft. wide and 7 ft. high is stoped from the back of the drift. Fig. 10 shows a cross-section and a longitudinal section of a typical gangway, giving the details of timbering above the gangway set for support of the filling. All gangways are 5 ft. by 9 ft. 10 in. in the clear, the distance from the sill to the top of the 10-in. by 12-in. sheeting being 13 feet.

Supervision of Drilling Operations.—On account of the difficult drilling ground at this mine, it has proved economical to place all drilling

operations under the supervision of one engineer. His duties may be enumerated briefly as follows: (1) To keep a detailed record of all drills underground and to see that they are kept in the best working condition; (2) to test new types of drills as they are received from the manufac-

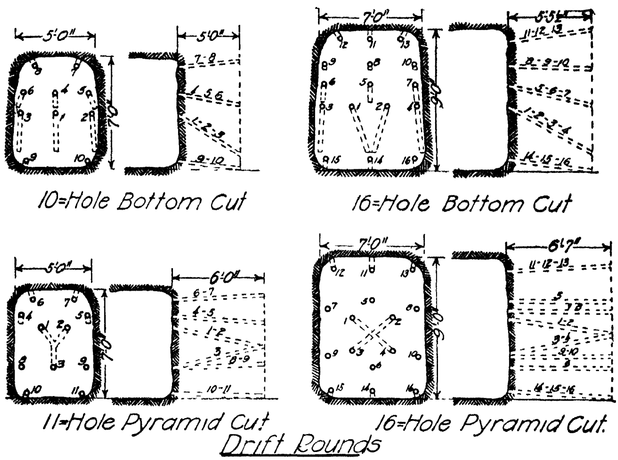


FIG. 5.—DRIFT ROUNDS.

turers; (3) to record and analyze the work accomplished by the drill bits; (4) to see that the type of round best adapted to the class of ground being drilled is used in all development work; and (5) to carry on experimental work relative to air pressures, hose, hose connections, valve designs, etc.

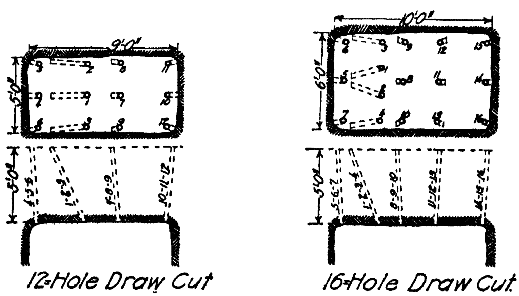
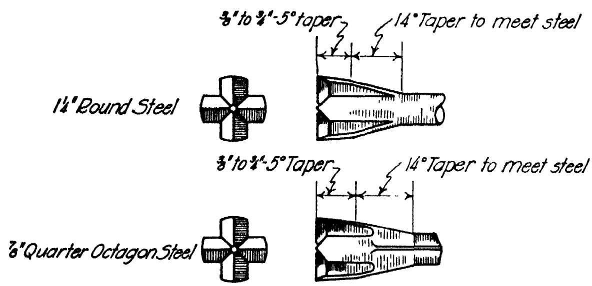


FIG. 6.—RAISE ROUNDS.

Drills Used.—Hammer drills of the 248 Ingersoll-Leyner type are used for all drifting and cross-cutting, as most of the ground on the lower levels is either tough quartz porphyry or massive sulfides. In ordinary ground one machine, mounted on a 3½ in. (9-cm.) vertical bar, is suffi-

cient for each drift. It is operated by one man, with the assistance of a shoveler for the set-up. In ground that requires two or more shifts to drill a round, two machines are mounted on the same bar and a chuck tender is provided to carry steel and powder for the two machines. The lighter types of mounted water drills have given satisfaction in black-schist and altered quartz-porphyry areas; these machines weigh approximately 100 lb. (45 kg.) unmounted and have a 2¼-in. cylinder. Every effort was made to substitute these light machines for the heavy Leyner drills in all classes of ground, but they were not successful. The greater drilling speed of the heavier drill in the hard porphyry and massive sulfides more than offsets the advantage of the one-man set-up with the lighter drill.

Types of Rounds.—Formerly the bottom cut was used exclusively, whereas in the present practice the pyramid cut is standard for all drift



Standard Changes

Steel	Short Starters	Long Starters	Seconds	Thirds	Fourths	Finishers	Long Finishers	Extra Long Finishers
Length 1¼" Rd.	12"-18"	18"-27"	27"-36"	36"-45"	45"-54"	54"-63"	63"-72"	Over 72"
Guage	1 15/16"	1⅞"	1 13/16"	1¾"	1 11/16"	1⅝"	1 9/16"	1½"
Length ⅞" Oct.	12"-18"	18"-30"	30"-42"	42"-54"	54"-66"	66"-78"	78"-90"	Over 90"

FIG. 7.—DOUBLE-TAPER CROSS-BIT.

rounds from a clean set-up. The bottom cut is still used whenever it is necessary to set up a cross-bar before the drift has been cleaned out, as it requires the drilling of only two or three holes from the lower set-up.

Drill Steel.—At the present time only two kinds of drill steel are used. These are 1¼-in. (3.2 cm.) hollow round steel for the heavy Leyner type of water drills, and ⅞-in. (2.2-cm.) hollow quarter-octagon steel for the light Jackhamer type and for the wet stoper drills. The size of bits is the same for both kinds of steel. Fig. 7 shows, in tabular form, the standard changes.

The breakage loss is less for 1-in. quarter-octagon steel than for ⅞-in. steel, but this advantage is offset by the greater drilling speed obtained with the ⅞-in. steel in drilling vertical holes. The ⅞-in. quarter-octagon

steel gives less breakage than either ⅞-in. hexagon steel or 1⅛-in. cruciform steel. Table 11 shows the steel breakage for December, 1917, and for a year later, December, 1918, after the adoption of a double-taper cross-bit and steel with quarter-octagon section.

TABLE 11.—*Steel Breakage*

	1¼-in. Round	⅞-in. Hexagon	1⅛-in. Cruciform	⅞-in. Quarter-octagon	Total
December, 1917					
Steel sharpened	19,227	10,482	4,779	2,978	37,466
Steel broken......... . .	578	427	296	62	1,363
Per cent. broken..	3.01	4 08	6.20	2.08	3.64
December, 1918*					
Steel sharpened....... .	28,121	4,256	405	11,471	44,253
Steel broken.....	252	77	34	190	553
Per cent. broken.......	0.89	1.78	8 40	1.66	1.25

* After introduction of double-taper cross-bits.

Types of Bits.—Experimental work carried on by H. W. Seamon, the efficiency engineer supervising all drilling operations, has resulted in the following improvements in the drill bits. When these tests were commenced, all Leyner machines used 1¼-in., hollow, round steel with cross-bits of 14° single taper. A 9-in. change in length and a ⅛-in. change in gage from a starter of 2⅜-in. to a finisher (eighth) of 1½-in., were the ordinary changes in use. On account of the rapid loss of gage with a single-taper bit, 1/16-in. changes in gage were unsatisfactory. The double-taper cross-bit, Fig. 7, and the double-taper arc bit were tested in comparison with the single-taper bit mentioned, using six 9-in. changes, with gage ranging from 2¼ in. to 1⅝ in. The results of these tests are given in Table 12.

TABLE 12.—*Comparison of Single- and Double-taper Bits*

Material Drilled	Average Drilling Speed, Inches per Min			Average Loss in Gage per Foot Drilled		
	Single-taper Cross-bit	Double-taper Cross-bit	Double-taper Arc Bit	Single-taper Cross-bit, Inch	Double-taper Cross-bit, Inch	Double-taper Arc Bit, Inch
Massive sulfides..	1.53	3.30	3.05	0.125	0 063	0 078
Jasper......... .	2 65	3.75	4.15	0.156	0 094	0.094
Porphyry.........	5.10	5.48	5.56	0.031	0 031	0 031

As 75 per cent. of development and stope drilling is in massive sulfides, the average increase of 105 per cent. in drilling speed in this formation by the use of the double-taper bit led to the prompt abandonment of the

single-taper bit. The theory of the double-taper bit, as developed by Geo. H. Gilman,[3] was fully demonstrated by the foregoing tests under most severe drilling conditions. With the slight difference in drilling speed between the double-taper arc bit and the double-taper cross-bit, the latter was adopted as the standard bit because it is simpler to make and easier to maintain. The tests also led to the adoption of a 1/16-in. (1.5 mm.) change in gage, as noted in Fig. 7. The loss in gage of the double-taper cross-bit was less than 1/16 in. per foot drilled in all ground except jasper. As jasper is encountered only occasionally in development work, this objection was not serious.

Table 13, derived in part from Table 12, shows the increase in drilling speed resulting from changes made in the drill steel, from single-taper cross-bits with 1/8-in. change of gage to double-taper cross-bits with a 1/16-in. change in gage. All holes bottomed at 1 5/8 in. after six 9-in. changes. The results obtained by drilling holes of different diameters

TABLE 13.—*Effect of Drill Bit Design on Drilling Speed*

	Average Drilling Speed, Inches per Minute			Per Cent. Increase in Drilling Speed 3 over 1
	1 Single-taper 1/8-in. Change	2 Double-taper 1/8-in. Change	3 Double-taper 1/16-in Change	
Massive sulfides..	1.53	3.30	4.20	174
Jasper..........	2.65	3.75	5.20	96
Porphyry......... .. .	5.10	5.48	7.88	54
Gage of starter, inches. . . .	2 1/4	2 1/4	1 15/16	

in the above classes of ground indicate that the drilling speed, in inches per minute, is closely proportional to the area excavated. The increased drilling speed obtained by using the double-taper bit with 1/16-in. change in gage was secured at a slight increase in the cost of sharpening per bit. Several other types of bits have been tried at the United Verde mine, but the cross-bits have given the best satisfaction.

Explosives.—Two types of high explosives, gelatin and ammonia dynamite, are in use. A 35-per cent. and a 50-per cent. strength of each type are used in the proportions here shown.

STRENGTH	PER CENT. OF TOTAL USED
35 per cent. ammonia dynamite....................	21
50 per cent. ammonia dynamite....................	29
35 per cent. gelatin dynamite......................	33
50 per cent. gelatin dynamite......................	17
Total..	100

[3] Drill Steel and Drill Bits for Metal Mining. *Eng. & Min. Jnl.* (May 12, 1917) **103**, 823.

As all holes in breast-stoping are horizontal, or nearly so, and the mine workings are generally dry, the conditions are favorable for the use of ammonia dynamite. Fifty per cent. of all blasting is done with ammonia dynamite, at 9 per cent. less cost than if gelatin dynamite were used. Gelatin dynamite is used in shaft sinking and for all raise work, as it is easier to load into vertical holes. The 50-per cent. strength of each type of explosive is used in the massive sulfides and jasper areas, and for all block-holing. The 35-per cent. strength is used for blasting in all black schist and porphyry.

The powder is stored in a concrete magazine located approximately 3500 ft. from the portal of the Hopewell (1000-ft. level) tunnel, which has a capacity of 2500 boxes, about 2 months' supply for the mine. Sufficient powder for each day is hauled into the mine on the graveyard shift and distributed to each level on the same shift. At the beginning of the day shift, all powder is removed from the shaft station to the powder magazine on each level. Keys to this magazine are held only by the powderman and the shift boss.

A smooth, black-finished, cotton-countered, safety fuse is now used instead of the triple-taped, gutta-percha fuse formerly in use, on account of the tendency of the latter to crack in cold weather. In shaft sinking, which is the only wet drilling in the mine, electric fuses are used. No. 8-X caps are standard for all work. All fuses are cut and capped by one man on the surface; a mechanical fuse-cutter clips 20 lengths of fuse at one time. Several spare blades are provided for this cutter so that a sharp blade may be used at all times. A cap crimper, operated by a foot pedal, rapidly and accurately crimps the caps on the fuse. With this equipment, a maximum of 800 fuses have been cut, capped, and crimped by one man in 60 min. The capped fuses are stored in waterproof cans for distribution each day to the working levels.

Blasting Crews.—In the spring of 1918, on account of the inexperience of many of the miners and the necessity of keeping a close record of all powder used, a blasting crew was organized. At this time there was a 3-hr. interval between the day and night shifts, and blasting was done on both shifts. When the blasting crew was organized, the day was divided into three shifts and all blasting, except block-holing, was left to the graveyard shift. All blasting is done by this crew of experienced miners under close supervision. Blasters are required to use tamping at all times. Little success was obtained in enforcing the use of tamping until this special crew was organized. On account of the large open stopes and the great number of development openings being driven at one time, this method is well adapted for use at this mine.

Representative Costs.—The data for a number of representative drifts and crosscuts driven in 1918, given in Table 14, show the wide variation in cost for development work in the different rock formations:

TABLE 14.—*Costs and Drilling Data on Representative Drifts and Crosscuts*

	Character of Rock Drilled		
	Massive Sulfides	Schist	Porphyry
Cost per foot			
Labor, miners	$12.39	$5.34	$2.54
Carmen and miscellaneous	5.69	3 96	3.58
Total labor	$18.08	$9 30	$6.12
Explosives	5.86	2.12	1.56
Supplies	0.40	0.28	0.20
Repairs, drills and steel	4 88	2.24	0 90
Compressed air	5.16	2 37	0 95
Total cost per foot	$34.38	$16.31	$9 73
Cross-section of drift, feet	6 × 8	6 × 8	6 × 8
Type of drills used	Ingersoll-248	Waugh-60	Ingersoll-2
Average depth of hole, feet	5 0	5 5	6 0
Number holes per round	16 0	16 0	11 0
Distance drilled per round, feet	80 0	88 0	66 0
Distance drilled per machine shift, feet	9 2	37 0	44 0
Advance per round, feet	4 0	4 4	5 4
Distance drilled per foot of advance, feet	20 0	20 0	12 2

Raises

The standard raise has a cross-section about 6 ft. by 11 ft. (1.8 by 3.3 m.) and is timbered with 5 in. by 8 in. (12.7 by 20.3 cm.) cribbing, as shown in Fig. 8. One compartment is single-lined with 2-in. (5-cm.) lagging (double-lined for stoping) and is used as an ore chute, while the other compartment is used as a manway, and for hoisting timber, steel, and explosives. This type of cribbed raise has been the standard at the United Verde mine for a number of years. It is interesting to note in this connection that the cribbed raise finally adopted as standard by the Copper Queen Consolidated Mining Co. at Bisbee, Ariz., and evolved independently by them,[4] differs but slightly from that shown in Fig. 8. The timber compartment is lagged in the Copper Queen set; at the United Verde mine this has not been considered necessary. If the ground is not too heavy or blocky, the cribbing is removed upon completion of the raise, and used in other raises before it is scrapped or cut into track ties.

A hand-rotated water stoper with 2¾-in. cylinder is used for all rais-

[4] Chas. A. Mitke: "Standardization of Mining Methods," 15, Fig. 15. N. Y., 1919. McGraw-Hill. From *Eng. & Min. Jnl.* of November and December, 1918.

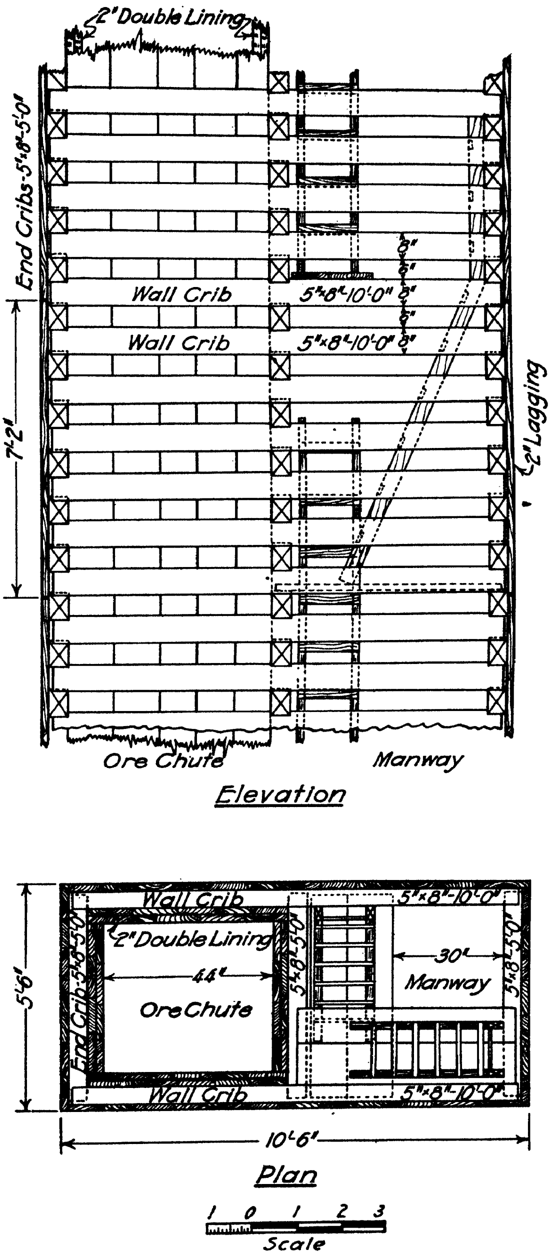


FIG. 8.—STANDARD CRIBBED CHUTE AND MANWAY.

ing in average ground. Due to the production of copper dust in drilling dry holes, with resultant ill effects upon the miner, no dry stopers are now in use. Stopers with automatic rotation have been tried with indifferent success. These machines are heavier than the hand-rotated stoper and require repairs much more frequently. This precludes their use at the United Verde mine at their present stage of development.

The standard type of round for raises is shown in Fig. 6. Either a 12-hole or a 16-hole draw-cut is generally used, depending on the nature of the ground drilled. In hard massive sulfides, 18 holes are usually required.

The data on representative raises driven during the year 1918, given in Table 15, shows the wide variation in cost per foot of advance in the different classes of ground.

TABLE 15.—*Cost and Drilling Data, Representative Raises*

	Character of Rock Drilled		
	Massive Sulfides	Schist	Quartz Porphyry
Cost per foot			
Labor, miners	$15.70	$6.27	$3.31
Miscellaneous	1.64	1.55	0.65
Total labor	$17.34	$7.82	$3.96
Explosives	5.28	3.26	0.99
Timber	2.05	1.63	0.92
Supplies	0.52	0.20	0.12
Repairs, drills and steel	4.35	1.79	0.63
Compressed air	4.60	1.90	0.66
Total cost per foot	$34.14	$16.60	$7.28
Cross-section of raise, feet	6 × 11	6 × 11	6 × 11
Drills used	Ingersoll-248	Ingersoll-CCW11	Ingersoll-CCW11
Depth of hole, feet	4.3	6.0	8.0
Number of holes per round	18.0	15.0	12.0
Distance drilled per round, feet	77.0	90.0	96.0
Dist. drilled per mach. shift, feet	11.0	28.0	48.0
Advance per round, feet	3.4	5.1	7.4
Distance drilled per foot of advance, feet	22.6	17.6	13.0

Bonus System on Development Work

A bonus system for the miners is applied on all raising, drifting, and shaft sinking. From cost and drilling data taken over a period of years, an average advance per machine shift in different classes of ground for a

fair day's work was determined. A series of bonuses ranging from 25 cents per shift upward has been formed from this base for each class of ground drilled. The miner and the company share equally in any decrease in cost that results from work better than the average. The miners naturally prefer a flat contract price to the bonus system. The contract system, however, is not feasible at the United Verde mine on account of the frequent changes in the formation and in the character of the rock encountered in development work.

There are two prerequisites to the successful operation of a bonus system in development work: (1) A fair and uniform rate must be made for each class of work; and (2) a rate once established must not be cut, no matter how advantageous it may prove to be to the miner. The first requirement is met by having all bonuses set by one engineer, whose familiarity and experience with the different classes of ground in the mine over a period of several years is such that he can quickly and fairly classify any new piece of development work. The second requirement is easily met by the fixed rule that the bonus engineer is permitted to increase the bonus rate where the ground proves harder than anticipated, but, under no conditions, may he cut a rate once established. The average miner is suspicious of any contract or bonus system, believing, from past experience, that rapid progress and high earnings for one month may result in a cut in his bonus rate for succeeding months. This suspicion can be allayed only by strict adherence to a fair policy, as outlined above.

General Exploration and Development Costs

Table 16 shows the average cost per foot for all development and exploration work during the year 1918. The total footage for the year 1918 was as follows: Drifts and crosscuts, 17,300 ft.; raises, 7716 ft.; total, 25,016 ft. Of this total, 14,778 ft. was driven for exploration and 10,238 ft. for development. The cost of compressed air used under ground was distributed between development and stoping on a basis of relative drill shifts worked. The cost of repairs, which includes all charges for drills and drill parts, drill steel and drill sharpening, is distributed on the same basis.

TABLE 16.—*General Exploration and Development Costs*

	Cost per Foot
Labor	$12.15
Explosives	3.33
Supplies	0 25
Timber	0.88
Compressed air	2.37
Repairs	2.24
Total	$21.22

Diamond Drilling

Diamond drilling at the United Verde mine serves two purposes; long holes are drilled to locate the different geological formations, and massive sulfide areas are developed and blocked out by numerous short holes. It has been found that the diamond-drill cores from schist areas cannot be relied on for accurate data on the mineralization. Consequently, the drill is used to locate and determine the size of schist areas, and the actual development is performed by drifts and crosscuts. The assays of cores from the massive sulfide areas have been found to check closely with subsequent drift development work. For this reason, and because of the high cost of drifting in such areas, exploration of the massive sulfide areas and the delimitation of the boundaries of large sulfide orebodies is usually done by diamond drilling. Fig. 1 shows the location of diamond-drill holes on a typical mine level.

The cost of diamond drilling varies with the class and character of formation, length and angle at which holes are drilled, and diameter of core removed. The ES bit, which gives a core 7/8 in. (2.2 cm.) in diameter, is used for most of the diamond drilling. On holes over 1000 ft. (304 m.) in length, the E bit (15/16 in. core) is used for the first 700 ft. of hole, and is then followed by the ES bit for the remainder of the hole.

The effect of pitch and total length of hole upon drilling costs is shown by Table 17, which was developed from cost data on numerous holes. The costs are based on the following unit costs of labor and carbon: Bit setter, per shift, $8.71; drill runner, per shift, $5.96; drill helper, per shift $5.46; first quality carbons, $110 per carat.

Table 17.—*Details of Diamond-drilling Costs, 7/8-in. Cores*

	Cost Per Foot
Horizontal holes, length 500 ft	$2 90
Holes pitching more than 5° from horizontal length 500 ft	$3.25
Horizontal hole, 1600 ft	$3.73

Cost of power is not included in any of the above costs.

Table 18.—*General Cost of Diamond Drilling*

	Cost Per Foot
Labor	$1.97
Supplies	0.40
Carbons	2.32
Compressed air	0.24
Repairs	0.06
Sub-total	$4.99
Cutting drill stations	0.12
Total	$5.11

The total diamond-drilling footage for the year 1918 was as follows: Drill holes, $\frac{7}{8}$ to $1\frac{5}{16}$-in. cores, 8373 ft.; drainage holes $2\frac{3}{16}$ in. diameter, 753; total, 9126 ft. Table 18 gives the detailed cost per foot for this drilling.

Stoping

The general conditions that govern the selection of a stoping method are stated, by Herbert Hoover,[5] to be: The dip, shape and size of orebodies, character of walls, cost of materials, and character of ore. The details of this broad generalization that are considered of prime importance in determining local stoping methods are: Necessity of safe and efficient working conditions for the men, fire hazard, complete extraction of ore, flexibility for stope development, ease of sorting out waste, and possibility of future mining of low-grade mineralized areas on the borders of present stopes.

The stoping methods and their relative importance are given in Table 19. The relative importance of the respective methods is rapidly

TABLE 19.—*Stoping Tonnage at United Verde Mine*, 1918

Method of Stoping	Dry Tonnage Mined	Per Cent of Total
Horizontal cut-and-fill	468,000	58.5
Incline cut-and-fill	11,200	1.4
Square-set and fill	68,000	8.5
Shrinkage and fill	117,000	14.6
Glory hole	136,000	17.0
Total	800,200	100.0

changing at the present time (December, 1919). Glory-hole mining will be abandoned in favor of open-cut steam-shovel mining early in the year 1920. No ground has been developed recently on the lower levels that will permit the use of shrinkage methods. Consequently, in future work, horizontal cut-and-fill and square-set will be the chief methods employed in underground mining, with subordinate use of the incline cut-and-fill method where applicable. A system of shrinkage stoping with pillar caving will be used for a converter orebody in quartz porphyry.

Selection and Comparison of Stoping Method.—On account of the relatively high copper content of the ore and the fact that it is smelted as it comes from the mine without concentration, no method can be employed that does not yield a practically complete extraction of the orebody, with-

[5] "Principles of Mining," 94. N. Y., 1909, McGraw-Hill Book Co.

out dilution by waste. This condition places any caving method out of consideration. The character and concentration of the orebodies also eliminates the possibility of using top-slicing methods.

Until the introduction of the horizontal cut-and-fill method under the superintendence of Robert E. Tally, practically all stoping was done by square-set and fill methods. Square-set stoping is well adapted to the United Verde orebodies except for the extreme fire-hazard involved, and the high cost of labor and timber. Compared with horizontal cut-and-fill methods, a square-set stope yields a much smaller daily tonnage, is more difficult to ventilate and supervise, and will not permit the accumulation of a reserve of broken ore. However, in many black-schist areas, and occasionally in highly mineralized sulfide stopes, the ground is slabby and so badly broken that it can be mined with safety only by the use of square-set stopes carried up in small sections. It is also necessary to use square-set methods whenever a horizontal cut-and-fill stope has been carried up to within 20 to 30 ft. (6 to 9 m.) of a filled stope on the level above.

Horizontal cut-and-fill methods overcome the objections to the square-set methods outlined. Both the square-set and the horizontal cut-and-fill methods have the following local advantages over the shrinkage and inclined cut-and-fill methods: (1) Flexibility, *i.e.*, ability to leave horses of barren or low-grade material unmined and to offset in ore when necessary as stoping progresses; (2) ease of prospecting the stope walls by crosscuts, which is particularly essential in schist stopes; and (3) ability to sort out low-grade rock or dike material cleanly and cheaply.

On account of the usual tendency of the ore to shatter well ahead of the ground blasted and the consequent danger from falling ground in working under a brow, incline cut-and-fill methods can seldom be used locally with maximum conditions of safety. The massive sulfides break in large pieces, which stand on a slope of 60° and entail much labor in block-holing and breaking with a hammer. The slight saving in mucking labor in an incline stope and the pronounced saving in handling waste fill are practically offset by the higher cost of handling timber and steel and the difficulty of erecting bulkheads, as compared with the horizontal method. As the latter method has the additional advantages outlined above, the incline cut-and-fill method is used only occasionally in narrow stopes where the walls are not strong enough to permit shrinkage stoping.

Shrinkage stopes are used only in hard ground in massive sulfide areas. The shrinkage method shows a lower cost for timbering and waste filling than any other method used and, in addition, permits the accumulation of a large reserve of broken ore. As the sulfide ore has a tendency to break in large masses, the cost of drawing shrinkage ore from the chutes on the level is greater than from other stopes where grizzlies are

used. Practically no sorting can be done to advantage in a shrinkage stope. In order to prevent any later movement of the stope walls, shrinkage stopes are filled as soon as possible after completion of the drawing. This precaution is necessary on account of the concentration of all orebodies within a relatively small area. The character of the United Verde orebody and its walls limits the application of this highly desirable method to an occasional stope.

Horizontal Cut-and-fill Stoping

The horizontal cut-and-fill method is used for mining some of the smallest orebodies in the mine as well as for those of greater area. Fig. 9 shows the application of this method to an orebody 70 ft. in width by 450 ft. (21 by 137 m.) in length, indicated in plan in Fig. 1.

Preliminary Development Work.—In present practice, the first step in the development of an orebody, after its position has been determined by a crosscut, is to drive drifts to ascertain its longitudinal extent. From these drifts, by means of short raises, crosscuts are driven on the second floor, 13 ft. (3.9 m.) above the level, at intervals of 100 ft. (30 m.) to determine the outline of the orebody. This work, if done on the level, would later interfere with the desired location of gangways and make the ground heavy around them.

The next step is to enlarge the drift to gangway size, about 8 ft. by 15 ft. in rock section, and to erect the gangway sets. The framing of gangway timbers, and the method of bridging over the gangways, Fig. 10, are designed to throw the entire weight upon the girts in the set. The 12 in. by 12 in. caps act chiefly as spreaders. Doubling-up posts and girts, 10 in. by 10 in. in section, are placed under the girts. If necessary, a solid row of these posts can be used to support the girt and bridging. This method does away with the necessity of using doubling-up posts under the caps, an undesirable method in that it seriously reduces the gangway clearance.

While the gangways are being timbered, ventilation raises, spaced approximately 200 ft. apart, are driven to the level above. These raises are located so that they may be used as waste raises in later stoping operations. Additional gangways are then driven 40 ft. apart and parallel to this first gangway. Chute pockets are located in every second set on alternate sides of the gangway, which gives a spacing of 11 ft. between them. Manways are located 100 ft. apart in each gangway. They should be offset from the gangway to avoid the possibility of a man stepping in front of an approaching ore train as he descends from a stope, and also to provide storage for steel and timbers when hoisting to the stope. Solid ground is left at the bottom of all ore chutes, and a chute mouth installed, as shown in Fig. 10.

Stope Silling.—When the gangways have been timbered and sheeted the stope is started on the second floor. From experience at this mine, it has been found advisable to start the sill floor 13 ft. above the level,

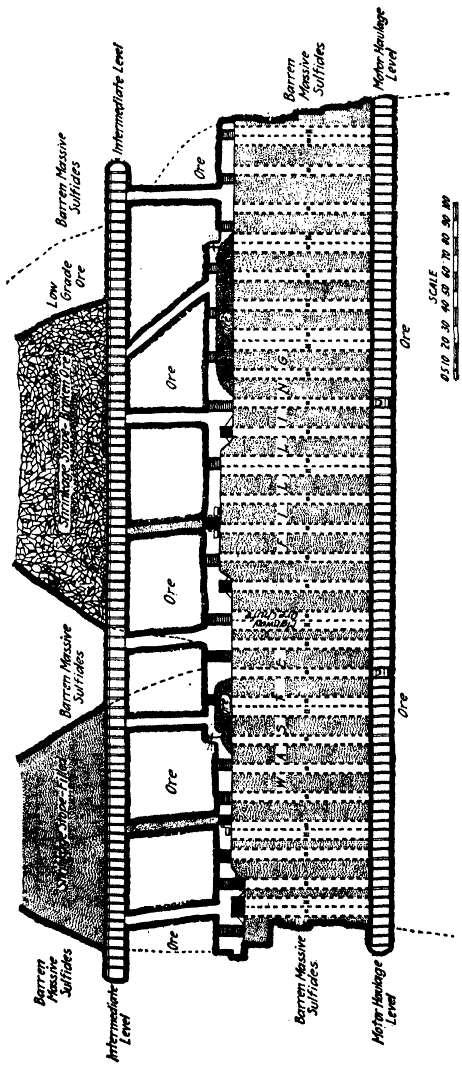


FIG. 9.—LONGITUDINAL SECTION—HORIZONTAL CUT-AND-FILL STOPE.

leaving the ground on the level intact except for the above-described development work. When a stope is silled on the floor of the level, the side thrust on the gangway timbers, transmitted through the waste

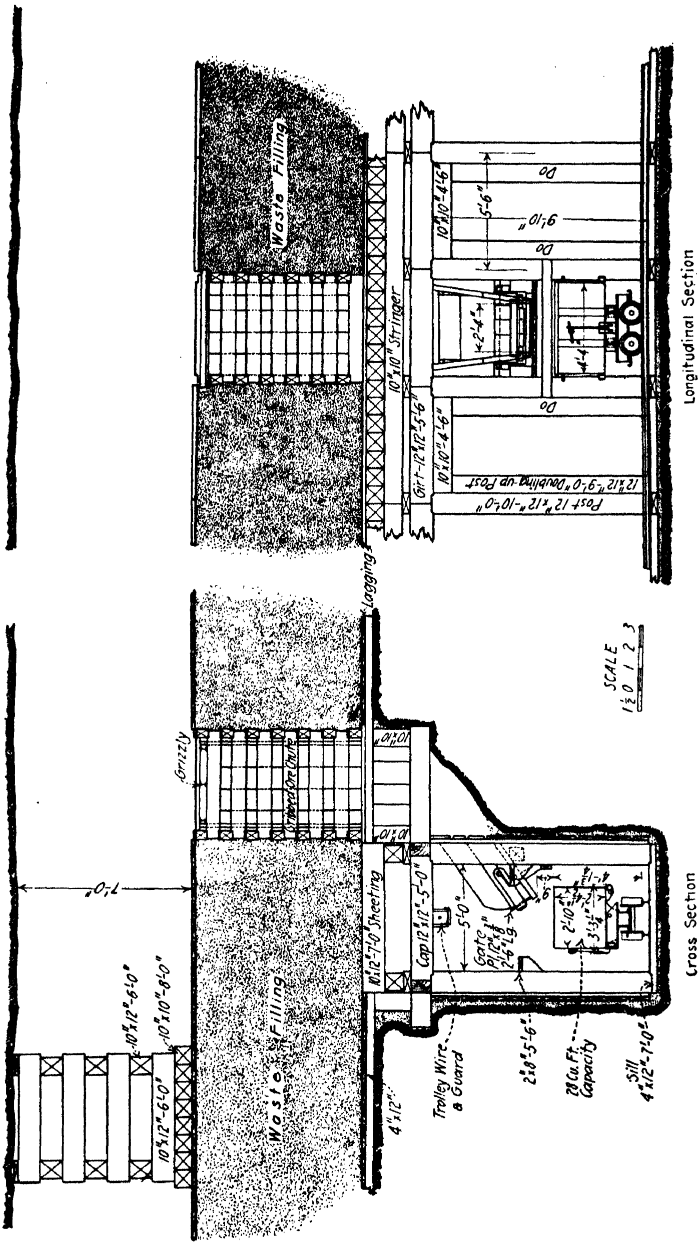


FIG. 10.—CHUTE AND GANGWAY TIMBERING.

filling, results in an excessive cost for maintaining gangway timbers. By leaving the sides of the gangway intact, the chief stress comes vertically upon the gangway. This is provided for by the timbering shown in Fig. 10; both methods of silling are represented on the geologic plan, Fig. 1.

A cut 7 to 9 ft. (2 to 2.7 m.) high is taken, using mounted hammer drills and horizontal holes. No air-feed stoper drills are used in these stopes, as vertical or steeply inclined holes produce a dangerous shattering of the back of the stope. As the silling proceeds, bulkheads of Oregon pine timber, 10 in. by 12 in. by 6 ft. long (0.25 by 0.3 by 1.8 m.) are erected with a maximum spacing of 25 ft. from center to center. Two-member bulkheads are used for average ground and three-member bulkheads, spaced as closely as efficient mucking conditions will permit, for heavy ground. In heavy ground, it is sometimes necessary to bridge between bulkheads with 10 in. by 10 in. timbers. Upon completion of the sill (second) floor, 4 in. by 12 in. bottom sills are laid and covered with 2 in. flooring, in preparation for a second cut of 7 feet.

Waste Raises.—Simultaneously with the silling, additional waste raises are driven. These raises are 6 ft. by 11 ft. in section and are spaced so that each raise will serve approximately 3000 sq. ft. of stoping area. The waste raises should be located close to either the foot wall or the hanging wall. Their location in the center of the stope weakens the back of the stope in the vicinity of each raise and adds unnecessary danger to the stoping operation.

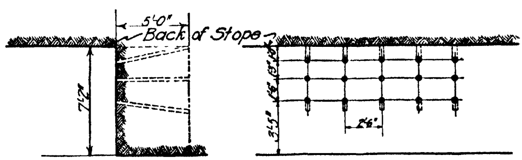
Routine of Stoping Operations.—After completion of the preliminary work outlined above, a second cut of 7 ft. is started from one of the waste raises. The weakest or heaviest point in the stope is selected for starting this cut, and the strongest ground is left, whenever possible, until the finish of the cut. Large hammer drills of the 248 Leyner type are used for all drilling in the massive sulfide stopes. In schist stopes, the lighter mounted water drills can be used to advantage. The types of rounds in use are shown in Fig. 11. In average ground two lines of 6-ft. holes are sufficient. In hard massive sulfide stopes, three lines of 5-ft. holes are required. The footage drilled per shift and the tonnage broken per shift depend on the class of ore being drilled, as may be seen by the following approximate figures.

Class of Ore	Average Footage per Drill Shift	Average Tonnage Broken per Drill Shift
Massive sulfides	24	21
Schist	48	40

In order to have a large reserve of broken ore available at all times, shoveling is not started in the stope until 2000 to 3000 tons of ore have been broken; thereafter shoveling and ore breaking proceed at the same pace.

It has been stated that the "brow" ahead of the drilling is the dangerous section of the stope. No shoveling is permitted under this brow, and the space for 20 ft. in advance of the brow is always kept roped off from the rest of the stope. Two roof inspectors, who are skilled miners with long underground experience, inspect the back of each stope and bar down any loose pieces of rock, each working on top of the broken ore ahead of the shovelers. Consequently, the back of each stope has had from five to twenty daily inspections before the shovelers begin their work under the 14-ft. back.

The shovelers work on the 2-in. (5 cm.) flooring, which serves the double purpose of keeping the broken ore and waste fill separate and providing an easy floor for shoveling. A No. 2 round-point shovel, made



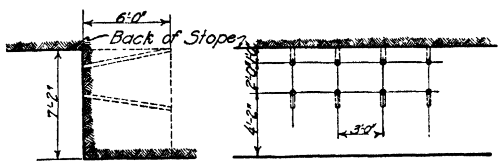


FIG. 11.—STOPE ROUNDS.

from chrome-nickel alloy steel, is the standard for all shoveling. The capacity of this shovel is 250 cu. in. (4096 cu. cm.) or 20.6 lb. (9.3 kg.) of average sulfide ore. The two methods of transfer to the ore chutes are direct shoveling into the ore chutes wherever possible, and by wheelbarrows for distances over 15 ft. (4.5 m.) The output per shoveler varies from 10 to 16 tons per shift, which includes the delays for breaking boulders to pass the grizzly openings. Barren country rock or dike material, when encountered in shoveling, is sorted and thrown back, to be covered by the advancing filling.

To prevent the dropping of large boulders into the 44-in. by 44-in. (111.7 cm.) chute opening, a grizzly made of 60-lb. (27 kg.) rails spaced to give two openings each 11 in. by 44 in., is built over each ore chute. This design is being superseded by one which provides four openings, each

15½ in. by 15½ in. This later design has the advantage of passing larger rounded pieces than the old-style grizzly and preventing the passage of long slabs, which now cause most of the delays by blocking at chute mouths and at the skip pockets.

Where the gangways are located on 40-ft. (12-m.) centers, ore chutes are spaced on approximately 20-ft. centers throughout the stope. By balancing the cost of shoveling against the cost of erection and maintenance of ore chutes, this spacing of chutes has been found to be the most economical. The ore chute is shown in section in Fig. 10, and is identical with that used for raising, except that double lining is required for all stope chutes. The standard chute mouth, shown in Fig. 10, is delivered underground cut to size.

Short bulkheads are built upon the broken ore, and bulkheads 14 ft. (4.2 m.) in height are erected from the shoveling floor to the back of the stope. As soon as the shovelers have provided sufficient cleared space, waste filling is dumped from the level above into the waste raise until no more of the filling will run into the stope. The cone formed at the bottom of the raise is then leveled off 7 ft. above the floor and a temporary chute is constructed. The filling is then distributed by tramming in cars of 18 cu. ft. (0.5 cu. m.) capacity, until the next waste raise becomes available. As the shovelers advance, the short bulkheads on the broken ore are taken down and 14-ft. bulkheads are built up from the floor of the stope. Similarly, as the filling advances, the 14-ft. bulkheads are removed and replaced by short bulkheads built upon the waste. When the ground is heavy, the long bulkheads are often necessarily left standing and the filling placed around them.

All bulkheads on filling are erected upon a mat of double 2-in. (5 cm.) lagging or 10-in. by 10-in. timber 8 ft. in length, in order to transmit the pressure to a large area of the filling. As soon as the cribbed ore chutes have served their purpose and are not needed by the shovelers, they are built up to the next floor and filling placed around them.

Table 20.—Horizontal Cut-and-fill Detailed Stoping Costs, 1918

Ore Breaking	Per Cent.		Cost per Ton
Labor, miners drilling	55.6	$0.210	
Miners block-holing	22.2	0 082	
Steel nippers and chucks	11.1	0.042	
Stope hoistman	11.1	0.042	
	100.0	$0.376	
Explosives		0.140	
Compressed air		0.116	
Repairs and replacements, steel and machines		0.154	
Total ore breaking			$0.786

Shoveling				
Shovelers			$0.440	
Shovels, sledges, wheelbarrows, etc			0.063	
Total shoveling				$0.503
Timbering				
Labor, erecting bulkheads	13.2		$0 0252	
Placing stope flooring and fencing pillars	13.4		0 0254	
Raising ore chutes and manways	15.2		0.0289	
Relining ore chutes and manways	45.3		0.0860	
Repairing chute mouths and gangway timbers	12.9		0 0245	
	100.0		$0 190	
Timber, bulkheads	14.3		$0.0258	
Chutes and manways, first erection	47.5		0.0859	
Chutes and manways, relining	23 3		0 0423	
Lagging, flooring (breakage and wastage)	3 1		0 0056	
Slabbing around chutes and manways	9 7		0 0176	
Fencing pillars	2.1		0 0038	
	100.0		$0.181	
Miscellaneous supplies			$0 013	
Total timbering				$0.384
Filling				
Cost of mining waste per ton of ore mined, labor		$0 017		
Explosives		0.011		
Compressed air		0.009		
Repairs		0 011	$0 048	
Cost of delivering waste to the stopes per ton of ore mined, labor		$0 252		
Electric power		0 001		
Repairs		0 001	$0.254	
Cost of distributing waste filling in stopes per ton of ore mined, labor			$0 274	
Total filling				$0.576
Total stoping cost				$2 249
Tramming Ore				
From stope chutes to skip pockets, labor		$0 204		
Electric power		0.003		
Repairs		0.003		$0 210
Total cost, stoping and tramming				**$2.459**

The filling is covered by a flooring of 2-in. lagging and is then ready for the next cut. This lagging is taken up on each floor as the filling advances and is used a second time as flooring, after which it is used for lagging the outside of the chutes to protect the cribbing, or as pillar lagging, to separate the filling from the ore pillars (40 ft. wide) left between large stopes.

In all stopes having an area of 10,000 sq. ft. (920 sq. m.) or more, it has been found economical to install an air hoist on the level with a light single-deck cage for handling men and supplies into the stope. In stopes smaller than 10,000 sq. ft. an Anaconda stope hoist is generally used The mounted-drill-column hoists are used almost exclusively for raise work on account of their much slower rope speed. The limited amount of timber and steel required for raise work and the advantage of portability and small amount of head room offsets the disadvantage of a slow rope speed.

When a horizontal cut-and-fill stope has been carried up to within 20 to 30 ft. of a level, the remaining pillar is mined in small sections by overhand square-set methods. The total area is divided into sections, and the weakest section is mined first.

In Table 20, are given the detailed costs of horizontal cut-and-fill stoping for the year 1918, when 468,000 dry tons were thus mined.

During 1918, about one-half of the total ore mined was hand trammed on the intermediate levels, in addition to the regular motor haulage on the main levels. The average cost for tramming per ton of ore mined was, therefore, much higher than actual tramming costs per ton, as given in detail under the heading Tramming. This cost, for 1918, was 21 cents per ton, with distribution as given above. The cost of ore breaking and shoveling in silling operations on the second floor is approximately 20 per cent. higher than the foregoing averages.

Timber Consumption.—The average consumption of timber in horizontal cut-and-fill stopes is as follows:

	Timber Consumption, Board Feet per Ton
Bulkheads	1.286
Chutes and manways, first erection	4.270
Chutes and manways, relining	2 096
Lagging, flooring (breakage and wastage)	0 277
Slabbing around chutes and manways	0.874
Fencing pillars	0 198
Total	9 000

Incline Cut-and-fill Stoping

The incline cut-and-fill method has been used on a few narrow orebodies, where one wall was too weak for the application of the shrinkage method and where there were no diorite dikes cutting through the ore-

body. In one of these orebodies, 35 ft. wide by 180 ft. long (10.6 by 54.8 m.) the operation was as follows: A drift was driven along the main axis of the orebody and the usual gangway timbering and chutes were erected. At the center of the orebody a waste and ventilation raise was driven to the level above. Inclined overhand cuts 7 ft. in width were then taken alternately on each side of the raise. The central location of the raise permitted steady operation of the stoping, one slope being

Table 21.—Incline Cut-and-fill Detailed Stoping Costs, 1918

Ore Breaking	Per Cent.		Cost per Ton	
Labor, miners drilling	59 3		$0.466	
Miners block-holing	25.9		0.204	
Steel nippers and chucks	14.8		0 116	
	100 0		$0 786	
Explosives			0.257	
Compressed air			0.289	
Repairs and replacements, steel and machines			0.384	
Total ore breaking				$1.716
Shoveling				
Shovelers			$0.185	
Shovels, sledges, etc			0.027	
Total shoveling				$0.212
Timbering				
Labor			$0.153	
Timber			0.148	
Supplies			0 011	
Total timbering				$0.312
Filling				
Cost of mining waste per ton of ore mined, labor		$0 017		
Explosives		0.011		
Compressed air		0.009		
Repairs		0.011	$0.048	
Cost of delivering waste fill to the stopes per ton of ore mined, labor		$0.252		
Electric power		0.001		
Repairs		0 001	$0.254	
Cost of distributing waste fill in stope per ton of ore mined, labor		$0.070	$0.070	
Total filling				$0.372
Total stoping cost				$2.612

filled with waste while breaking ore was in progress on the other. When this work was started, it was thought that all ore would roll by gravity to the chutes at the bottom of the incline as soon as a sufficient height was reached in the stoping. This did not occur, however, as the large blocks of broken ore stood on a slope of about 60°, without a tendency to roll. It was therefore necessary to carry intermediate ore chutes up through the filling, in addition to the regular chutes. It was also extremely difficult to avoid the admixture of ore and waste, although flooring was used on the filling.

In Table 21 are given the detailed costs of incline cut-and-fill stoping for the year 1918, when 11,200 dry tons were mined by that method.

It should be noted in reference to these costs that all incline cut-and-fill stopes are in exceptionally hard massive sulfides, which accounts for the high cost of ore breaking.

Square-set and Fill Stoping

The overhand square-set stoping method in use at the United Verde mine differs but little from standard practice for this method. Gangways are timbered in the same manner as for a horizontal cut-and-fill stope. All sets are 5 ft. 6 in. (1.67 m.) square and 7 ft. 2 in. (2.2. m.) high. The regular square-set chute and manway have the same dimensions as the cribbed chute and manway shown in Fig. 8.

The square-set method is used only in mining heavy ground, in coming up under completed stopes, and for stope pillars, or wherever it is necessary to keep the minimum amount of ground open. Consequently, only a few sets more than the mining floor are left unfilled at any time. This condition prevents the advantageous use of slides and necessitates the shoveling of practically all the broken ore into chutes spaced 22 ft. from center to center.

In Table 22 are given the detailed costs for square-set and fill stoping for the year 1918, when 68,000 dry tons were mined in that way.

Shrinkage-and-fill Stoping

This method is applicable to the mining of orebodies of massive sulfides when the walls consist chiefly of low-grade sulfides. The development and sill-floor work for shrinkage stoping is similar to that described in detail for the horizontal cut-and-fill method. After the sill floor has been mined, ore breaking proceeds in 7-ft. cuts, as in horizontal cut-and-fill work. As many boulders as possible are block-holed after each round, to avoid the costly blasting in the chutes upon the level. Cribbed chutes and manways spaced 100 ft. apart are used for ventilation and supplies. Occasionally 10-in. by 12-in. bulkheads are built upon the

TABLE 22.—*Square-set and Fill Detailed Stoping Costs*, 1918

Ore Breaking	Per Cent.		Cost per Ton
Labor, miners drilling	70.2	$0.280	
Miners block-holing	15.0	0.060	
Steel nippers and chucks	14.8	0.059	
	100.0	$0.399	
Explosives		0 085	
Compressed air		0.138	
Repairs and replacements, steel and machine		0.184	
Total ore breaking			$0.806
Shoveling			
Shovelers		$0 479	
Shovels, sledges, etc		0 046	
Total shoveling			$0 525
Timbering			
Labor		$0 582	
Timber		0 612	
Supplies		0 046	
Total timbering			$1.240
Filling			
Cost of mining waste per ton of ore mined, labor		$0 017	
Explosives		0 011	
Compressed air		0 009	
Repairs		0.011	
			$0.048
Cost of delivering waste to the stopes per ton of ore mined, labor		$0.252	
Electric power		0 001	
Supplies		0.001	
			$0.254
Cost of distributing waste filling in stopes per ton of ore mined, labor			$0.247
Total filling			$0.549
Total cost of stoping			$3.120

broken ore where blocky ground is encountered. No other timber, except as noted above, is used in this method. On account of its many advantages and low cost, this method is used wherever practicable.

In Table 23 are given the detailed costs for shrinkage stoping for the year 1918, when 117,000 dry tons were mined in that manner.

TABLE 23.—*Shrinkage Stoping, Detailed Costs,* 1918

Ore Breaking	Per Cent.			Cost per Ton
Labor, miners drilling	60.0		$0.428	
Miners block-holing	25.0		0.178	
Steel nippers and chucks	15.0		0.107	
	100.0		$0.713	
Explosives			0.247	
Compressed air			0.251	
Repairs and replacements, steel and machines			0.334	
Total ore breaking				$1.545
Shoveling				
Shovelers			$0.015	
Shovels, sledges, etc.			0.022	
Total shoveling				$0 037
Timbering				
Labor			$0 008	
Timber			0.079	
Supplies			0 006	
Total timbering				$0.093
Filling				
Cost of mining waste per ton of ore mined, labor		$0.017		
Explosives		0 011		
Compressed air		0.009		
Repairs		0.011	$0.048	
Cost of delivering waste to stopes per ton of ore mined, labor		$0.252		
Electric power		0 001		
Supplies		0.001	$0.254	
Cost of distributing filling in stopes per ton of ore mined, labor			$0.000	
Total filling				$0.302
Total cost of stoping				$1.977

Shrinkage Stoping with Pillar Caving

To meet the smelter requirements for free silica, an orebody in quartz-porphyry schist carrying 1.5 per cent. copper as secondary chalcocite is being prepared for mining. This orebody contains approximately 1,000,000 tons and extends from the 160-ft. to the 600-ft. level. It is roughly pear-shaped in outline, and is about 250 ft. (76 m.) long and 150 ft. (45 m.) wide. The gradation from 1.5 per cent. in copper content to barren porphyry is gradual on all sides of the orebody.

On the 300-ft. and 400-ft. levels, small blocks of this ore have been mined by shrinkage stoping and the stopes left filled with broken ore. Repair costs for gangway timbering, due to the heavy ground encountered, were excessive for the small tonnage obtained. It is proposed to mine this orebody by a combination of the Ray system of narrow shrinkage stopes and pillar caving with the Inspiration method of ore drawing through branched raises. At the present time, motor haulage gangways have been completed on the 600-ft. level, and grizzly drifts are being driven 40 ft. above this level.

TABLE 24.—*Glory-hole Stoping, Detailed Costs,* 1918

Ore Breaking	Per Cent.		Cost per Ton
Labor, miners drilling and block-holing	91.6	$0 172	
Steel nippers and chucks	8.4	0.016	
	100.0	$0 188	
Explosives		0 118	
Compressed air		0.054	
Repairs and replacements, steel, and machines		0.072	
Total ore breaking			$0.432
Shoveling			
Shovelers		0 026	
Shovels, sledges, etc		0 022	
Total shoveling			$0.048
Timbering			
Labor		$0 032	
Timber		0.011	
Supplies		0.001	
Total timbering			$0.044
Mining Waste			
Labor		$0 036	
Explosives		0 030	
Compressed air		0 014	
Repairs		0 018	
Total mining waste			$0.098
Tramming Waste			
Labor		$0.046	
Electric power		0 000	
Repairs		0 000	
Total tramming waste			$0.046
Total stoping cost			$0 668

The costs for mining and tramming waste are for handling the 20 per cent. of waste encountered in glory-hole stoping.

Glory-hole Stoping

Seventeen per cent. of the total mine tonnage for the year 1918 was obtained from glory-hole mining above the 160-ft. level. Raises were driven 80 to 150 ft. to the surface from gangways located on the 160-ft. level. These raises were located approximately 60 ft. apart and served as starting points for underhand stoping with unmounted hammer drills. Twenty per cent. of the total glory-hole area has proved to be either unmineralized or too low-grade to be mined as ore. A sufficient number of mill holes were used to maintain the desired tonnage of ore and at the same time permit the separate mining of these low-grade and waste areas. All ore left in and around this glory hole will be mined later by steam-shovel operations.

In Table 24 are given the detailed costs of glory-hole stoping for the year 1918, when 136,000 dry tons were obtained from that source.

TABLE 25.—*Cost of Waste Fill*, 1918

		Cost per Ton
Mining cost at the 300 glory hole:		
Labor	$0 0365	
Explosives	0.0225	
Compressed air	0.0187	
Repairs	0 0246	$0 1023
Tramming:		
Motor haulage, from glory hole to waste raise *A* in mine; distance 2500 ft	$0.1360	
Motor haulage, from waste raise *A* to waste raise *B* on the 1000-ft. level for transfer to stopes below this level; distance 800 ft	0 1500	
Hand tramming, from waste raise *B* on an intermediate level to the stopes; average distance 350 ft	0 2240	
Bypassing waste from level to level	0.0224	$0.5324
Total cost for 1 ton of filling at stopes		$0 6347

One ton of waste as filling is equivalent to about 2.10 tons of average ore mined; the above costs per ton of ore mined are then as follows:

		Cost per Ton
Cost for mining waste per ton of ore mined:		
Labor	$0.017	
Explosives	0.011	
Compressed air	0.009	
Repairs	0 011	$0.048
Cost for tramming and delivery to the stopes:		
Labor	$0 252	
Power	0.001	
Repairs	0 001	$0.254
Cost of filling at the stopes per ton of ore mined		$0.302

Mining of Waste for Stope Filling

A main waste raise, 9 ft. by 20 ft. (2.7 by 6 m.) in section, extends from the surface to the 1800-ft. level, with drawing-off chutes on each intermediate level. A grizzly and blasting chamber are located at an offset in the raise about 50 ft. above the 500-ft. level. A station has been cut on this level for the installation of a 48-in. by 36-in. (122 by 91 cm.) Blake crusher, which will reduce all coarse waste to 6-in. (15 cm.) size. This crushing should greatly reduce the cost for bypassing from level to level, and for loading into cars on the levels and in the stopes. Fine waste also packs more tightly than coarse, therefore the filled stopes will be left in much better condition than at present. This is important, as it may later be found advisable to mine the low-grade material on the walls of the present stopes.

Prior to 1919, all waste for filling, except development waste, was mined in a crushed limestone area 2000 ft. distant from the general mine workings. The mining cost was low, but it was necessary to make two transfers of the waste in the mine before it could be delivered to the stopes below the 1000-ft. level. The final cost was very high, as is shown by the following comparison with present costs when using the main waste raise just described. The cost of mining and delivering waste filling to the stopes below the 1000-ft. level is given in Tables 25 and 26.

TABLE 26.—*Present Cost of Filling at Stopes*, 1920

		Cost per Ton
Mining cost at the main waste raise:		
Labor	$0.10	
Explosives	0.05	
Compressed air	0.02	
Repairs	0.03	$0 200
Storage-battery motor haulage from main waste raise to stopes		0.154
Bypassing waste from level to level		0.020
Crushing coarse waste to 6-in. size		0.060
Total cost per ton of filling delivered to stopes		$0.434
Cost of filling at the stopes, per ton of ore mined		$0.207

Recapitulation of Stoping Costs

Table 27 gives a summary of stoping costs as detailed under each method in the foregoing pages.

STEAM-SHOVEL MINING

Most of the orebodies above the 500-ft. level are at present bulkheaded and inaccessible on account of mine fires. As the ground in this area was generally badly fractured, square-set methods were necessarily used. By the use of the plenum system of forcing air under pressure into

TABLE 27.—*Summary of Stoping Costs in* 1918

	Horizontal Cut-and-fill	Incline Cut-and-fill	Square-set and Fill	Shrinkage and Fill	Glory-hole	Totals
Dry tons stoped	468,000	11,200	68,000	117,000	136,000	800,200
Per cent. of total stoping tonnage	58.5	1.4	8 5	14.6	17.0	100.0
			Costs per Ton Stoped			
Ore breaking, labor	$0.376	$0.786	$0.399	$0.713	$0.188	$0.401
Explosives	0 140	0.257	0.085	0.247	0 118	0 149
Compressed air	0 116	0.289	0 138	0 251	0 054	0 129
Repairs	0 154	0.384	0 184	0.334	0.072	0.172
Total	$0 786	$1.716	$0.806	$1.545	$0 432	$0.851
Shoveling, labor	0 440	0.185	0.479	0 015	0 026	0 308
Supplies	0 063	0 027	0 046	0 022	0.022	0 048
Total	$0.503	$0.212	$0.525	$0 037	$0.048	$0 356
Filling in stope, labor	0.274	0 070	0.247			0 182
Timbering, labor	0 190	0 153	0 582	0 008	0 032	0 169
Timber	0.181	0.148	0 612	0 079	0 011	0 173
Supplies	0.013	0.011	0 046	0 006	0 001	0 013
Total	$0.384	$0 312	$1 240	$0.093	$0 044	$0.355
Mining waste,[a] labor	0.017	0 017	0 017	0 017	0.036	0 020
Explosives	0 011	0 011	0 011	0 011	0 030	0 014
Compressed air	0 009	0.009	0 009	0 009	0.014	0 010
Repairs	0 011	0 011	0 011	0 011	0 018	0 012
Total	$0 048	$0.048	$0.048	$0 048	$0 098	$0 056
Delivering waste filling, labor	0 252	0 252	0 252	0 252	0 046	0 217
Power	0 001	0 001	0 001	0 001	0 000	0 001
Repairs	0 001	0 001	0 001	0 001	0 000	0 001
Total	$0 254	$0 254	$0.254	$0 254	$0 046	$0 219
Total stoping, labor	1 549	1 463	1 976	1 005	0 328	1 297
Explosives	0 151	0 268	0 096	0.258	0 148	0 163
Timber	0.181	0 148	0.612	0.079	0 011	0 173
Compressed air	0.125	0.298	0 147	0 260	0.068	0 139
Electric power	0.001	0.001	0 001	0.001		0 001
Repairs	0.166	0 396	0.196	0 346	0 090	0.185
Supplies	0 076	0.038	0 092	0 028	0.023	0.061
Total	$2.249	$2.612	$3.120	$1.977	$0.668	$2.019
Tramming ore,[b] labor	0.204	0.204	0.204	0.204	0.204	0 204
Electric power	0.003	0.003	0.003	0.003	0 003	0 003
Repairs	0.003	0.003	0 003	0.003	0.003	0 003
Total	$0.210	$0.210	$0.210	$0.210	$0.210	$0.210
Total stoping and tramming	$2.459	$2.822	$3.330	$2.187	$0.878	$2.229

Cost per ton of ore delivered to smelter in 1918 (861,250 tons) $2.070

[a] Distributed uniformly against all methods except glory-hole.

[b] Distributed uniformly against all methods.

the fire stopes, it was found possible to mine this ore, but only at an extremely high cost. The present plan, now under way, is to strip a total of 14,000,000 cu. yd. (10,796,000 cu. m.) overburden and complete the mining of all the orebodies of the mine to the 400-ft. level by open-pit steam-shovel methods. It is estimated that approximately 5,000,000 tons of direct-smelting ore will be recovered to this depth. This is equivalent to 1 ton of ore recovered for each 2.8 cu. yd. of overburden removed.

Tramming

Trolley type of motor haulage is used on the 1000-ft., 1200-ft., 1500-ft., and 1800-ft. levels, the present main haulage levels of the mine, as well as in several of the upper levels. Hand tramming is used on the 1350-ft. and 1950-ft. levels; on the 1650-ft. level, storage-battery haulage was recently installed. All ore on these last three intermediate levels is trammed a maximum distance of 300 ft. to transfer raises which connect with the main haulage levels.

Cars and Tracks

The size of car used for all hand tramming and motor haulage was limited, until recently, by the size of the cages used in No. 3 shaft. This car is a typical side-dump type with a capacity of 18 cu. ft. (0.5 cu. m.). It can be dumped without uncoupling from the train and without stopping the train. With the introduction of skip hoisting, a car of the same type with a capacity of 28 cu. ft. has been designed. The dimensions of these two cars compare as follows:

	Inside Dimensions in Inches			Overall Dimensions in Inches		
	Length	Width	Depth	Length	Width	Height Above Rail
18-cu. ft. car	44	30	24	49	35	45
28-cu. ft. car	52	34	28	59	39	50

Total weight, including trucks, of 18-cu. ft. car is 960 lb.; of 28-cu. ft. car, 1600 lb.

Two types of car wheels are used, one a 12-in. (30 cm.) manganese-steel wheel with Hyatt roller bearings, and the second a 12-in. wheel with chilled cast-iron tread and solid roller bearings of the Sanford-Day type. Both types of bearings have given excellent satisfaction. Manganese-steel wheels are preferable for the severe service in motor haulage. The wheel tread has a minimum width of 3 in. to avoid derailment at curves and switches.

The use of 16-lb. (7.25 kg.) rails, ties 4 in. by 6 in. by 3 ft. (10 cm. by 15 cm. by 0.9 m.) and 18-in. gage is standard throughout the mine for all hand tramming. Motor haulage levels are equipped with 35-lb. rails, ties 6 in. by 6 in. by 3 ft. and 18-in. gage. Stub switches are used for both sizes of rails. Manganese-steel frogs are used on all motor haulage main-line switches. On main haulage levels curves are planned with a minimum radius of 50 feet.

Haulage Systems

The 250-volt direct current for motor haulage is carried by No. 0 wire, except in the Hopewell tunnel, where No. 000 is used. Wherever the wire is less than 8 ft. (2.4 m.) above the rail, it is guarded on each side by 2-in. by 8-in. (5 by 20 cm.) boards. A hinged board, 5 ft. in length and fastened to one guard, is provided at all chutes (Fig. 10). When hooked to the opposite guard by the trammer when loading cars, this arrangement completely covers the live trolley wire, preventing accidents. Pin-terminal bonds are used on all trolley installations.

Six-ton, Jeffrey, trolley-type mine locomotives, with solid slab frames are the standard in use for all main haulage levels. These locomotives are provided with two 18-hp. motors, which are connected to the drive wheels by intermediate gearing on account of the narrow gage of the track. Steel-tired wheels and ball-bearing armatures aid in maintaining a low upkeep cost. The rated drawbar pull is 3000 lb. at a speed of 6 mi. per hr. on level track. A train of twenty cars is used for ordinary haulage, which is a trailing load of 35 tons when using 18-cu. ft. cars and hauling massive sulfide ore.

A 3-ton, Baldwin-Westinghouse storage-battery locomotive equipped with eighty Edison type A-4 cells was recently placed in operation on the 1650-ft. (intermediate) 'evel. The primary object of this installation was to replace hand tramming of waste against a 0.4-per cent. grade. This motor has a drawbar pull of 800 lb. at 3½ mi. per hr., a speed sufficient for the short haul through timbered gangways. The batteries are changed and given a boosting charge each shift, using the 250-volt direct-current mine service line at the shaft station. The batteries are arranged in three crated sections for convenience in handling. This storage-battery locomotive gave such satisfactory service at such low costs, compared with the hand tramming, that duplicate equipment will soon be installed on the 1350-ft. and 1950-ft. intermediate levels.

Ore from the stopes on the 1650-ft. level is trammed an average distance of 200 ft. to transfer raises connecting with the 1800-ft. haulage level. An average of 44 tons per shift is handled by hand tramming at a cost of 12.7 cents per ton with trammers' wages at $5.60 per shift. The total cost per ton for storage-battery motor haulage, handling 215 tons per shift is as follows:

Labor (one motorman and two loaders)	$0.084
Power (25 kw.-hr. per shift)	0.004
Depreciation on motor equipment	0.006
Repairs and inspection	0.002
Total	$0.096

Three-fourths of the chutes in the above service have a capacity for only two or three cars, as they draw from stopes where silling operations are in progress on the second floor. The tonnage handled per shift by motor haulage should be increased 30 per cent. when these stopes have been carried up a few floors, and the chute capacity correspondingly increased. In hauling waste to the stopes an average distance of 350 ft. up a 0.4-per cent. grade, the reduction in cost by the storage-battery motor was much greater. The comparative costs were as follows; all of the costs include loading and dumping:

	Tons per Shift	Average Cost per Ton
Hand tramming	25	$0.224
Storage-battery motor haulage	135	$0.153

Cost of Trolleg-type Motor Haulage: During the months of October and November, 1919, 65,185 tons were loaded, trammed an average distance of 850 ft. (259 m.) and dumped into skip pockets at No. 5 shaft on the 1800-ft. level at the following costs per ton:

		Cost per Ton
Labor, motormen	$0.0357	
Switchmen	0.0328	
Grizzly man at skip pockets	0 0157	$0.0842
Power		0.0059
Depreciation, motor equipment		0 0064
Repairs to motors and supplies		0 0034
Maintenance of cars and track, estimated		0.0080
Total cost per ton carried		$0 1079
Total cost per ton-mile		0 6700

The following record of an average trip shows that only 24.9 per cent. of the total time is used in actual haulage and switching. The cost for actual hauling is $0.0349 per ton carried, or, 21.7 cents per ton-mile, including the return trip.

Operation	Time Consumed, Minutes	Per Cent. of Total Time
Loading and gathering cars	31.00	63.4
Hauling, loaded trip	2.75	5.6
Dumping cars	5.75	11 7
Hauling, empty trip	3.25	6 6
Switching cars	6.25	12.7
Total	49.00	100.0

All of these costs were obtained with a car of 18 cu. ft. (0.5 cu. m.) capacity. Decreased costs for loading and haulage should be obtained with the 28-cu. ft. car.

Pumping and Drainage

The average flow of water in the mine during the months of September and October, 1918, was as follows:

Copper-bearing water above 1000-ft. level, gallons per minute.....	17
Barren water above 1000-ft. level, gallons per minute............	36
Barren water below 1000-ft. level, gallons per minute............	52
Total, gallons per minute	105

The amount of water flowing above the 1000-ft. level is greatly in excess of these figures during periods of heavy rains or snows. The water below the 1000-ft. level, which must be handled by the pumps, seldom exceeds 60 gal. (227 l.) per min. The pumping problem at the United Verde mine has, therefore, been a simple one. A 50,000-gal. sump has been provided on the 1950-ft. level by driving the shaft crosscut 5 ft. (1.5 m.) below grade for a distance of 300 ft., and then concreting this section to track grade. This sump provides storage for 16 hr. under ordinary conditions.

At present, the water is handled by air pumps in two lifts, from the 1950-ft. to the 1500-ft. level, and from the 1500-ft. to the 1000-ft. haulage level. The air pump on the 1950-ft. level is a single-acting plunger type, 32 in. by 10 in. by 24 in. (80 by 25 by 60 cm.) stroke, with a capacity of 300 gal. (1135 l.) per min. at 90 lb. (40.8 kg.) air pressure. The air pump on the 1500-ft. level is a duplex, plunger type, 10 in. by 8 in. by 24 in. stroke, with a capacity of 230 gal. per min. at 90 lb. air pressure. The pump column is a 6-in. copper-wound pipe, designed for a head of 500 feet.

The air pump on the 1950-ft. level will soon be replaced by an Aldrich quintuplex pump, 6½ in. by 12 in., direct-connected to a 100-hp., 430-r.p.m., 2200-volt slip-ring induction motor. This pump has a capacity of 500 gal. per min. under 1000-ft. head, and will pump directly to the 1000-ft. haulage level upon completion of the installation of an 8-in. lead-lined pump column in No. 5 shaft. All pumps are fitted with bronze rods, bronze-lined pistons, and bronze-lined stuffingbox glands to resist the action of the mine water, which carries a small amount of copper in solution.

A considerable amount of concrete ditch work is required to prevent seepage of mine water through mineralized schist areas. The seepage of water through these areas, or along the diorite dikes within orebodies, makes the ground heavy and dangerous for mining. The small cost of

ditch work is many times repaid by the greater safety and improved mining conditions in the stopes below. Diamond-drill drainage holes $2\frac{3}{16}$ or $2\frac{13}{16}$ in. in diameter, located as shown in Fig. 1, are used to keep the mine water well away from the large stopes of the lower levels.

Mine Fires

Mine fires at the United Verde mine and the methods for their prevention and control have already been discussed in detail by Robert E. Tally.[6] At the present time all fire stopes are sealed off from the other workings by concrete bulkheads, as it is thought that it will be more economical to mine the fire areas by open-cut steam-shovel methods than to attempt to work them by the plenum system.

Ventilation

There have been few changes in the methods of ventilation at the United Verde mine since they were described by Robert E. Tally.[7] On account of the rapidly increasing development of the lower mine levels, it has been necessary to install a No. 15, 90-in. (229-cm.), single-inlet, Sirocco pressure fan of 175,000 cu. ft. (4900 cu. m.) capacity. This fan is located on the 1000-ft. level, 6300 ft. (1920 m.) from the mouth of the Hopewell haulage tunnel. This long intake tunnel involves a considerable friction loss, but has the advantage of cooling or warming the outside air so that air of uniform temperature is delivered to the fan. Typical temperature readings noted below very clearly show this condition.

Date	Temperature, Degrees F.	
	Mouth of Hopewell Tunnel	At No. 15 Fan Inlet
June 5, 1919	76	57
Nov. 28, 1919	35	54

The intake tunnel for this central fan is 120 sq. ft. (11 sq. m.) in cross-section. The air from the fan is delivered under 4 in. (10 cm.) water-gage pressure to a ventilation drift of the same cross-section, and then splits in drifts and raises of considerably greater total section for general distribution to the mine workings. Under present conditions, the air is forced downward through the working stopes and outlets to the shafts on the lower levels. As the tendency of the warm air is to rise, a more efficient method, wherever practicable, would be to force the air

[6] Mine Fire Methods Employed by the United Verde Copper Co. *Trans.* (1916) **55**, 186.

[7] *Op. cit.*, 192.

directly to the lowest mine level and allow it to work upward through the stopes.

Typical stope conditions before and after the installation of the large fan were as follows:

Using	Temperature, Degrees F.	Humidity, Per Cent.	Velocity, Ft. per Min.	Volume, Cu. Ft. per Min.
No. 8 fan	80	100	25	60
No. 15 fan	70	82	150	360

PRECIPITATION

Most of the copper-bearing mine water is formed during the progress of the natural surface drainage through the oxidized zone and through old filled stopes above the 500-ft. level. The amount formed varies with the surface precipitation. The total flow during the dry season is about 15 gal. (56 l.) per min., and in wet periods increases to about 50 gal. per min. The average flow is approximately 25 gal. per min. This water was formerly treated by three separate plants, located near the portals of the 300-ft., 500-ft. and 1000-ft. tunnels. Comparative data on these three plants, based on records for the years 1916, 1917 and 1918, are given in Tables 28 and 29. Of the total flow treated at the 1000-ft. level flumes, about 100 gal. per min. was barren mine water. The effect of this dilution and fouling of the copper water is seen in the low recovery and the poor quality of the precipitate that was produced.

TABLE 28.—*Data on Precipitation Plants*

	300-ft. Level Flumes	500-ft. Level Flumes	1000-ft. Level Flumes	Combined Plants
Total length of flumes, feet	650 5	1914	2735	5299
Net fall of flumes, feet	5 0	16.3	19 08	40 38
Average grade of flumes, per cent	0 77	0 85	0 70	0.76
Average width of flumes, feet	3.5	4.8	5 0	4 78
Total precipitating area, square feet	2277	9187	13,675	25,139
Normal flow, gallons per minute (1918)	7 5–15	7 5–25	100–150	115–190
Average assay, intake, per cent. (1918)	0 0302	0 176	0 0088	0 0216
Grains per gallon (1918)	17.7	102 8	5 15	12.6
Average assay, discharge, per cent. (1918)	0 0055	0 0027	0 0031	0 0032
Grains per gallon (1918)	3.2	1 6	1 81	1 87
Precipitate production, tons	128.9	759.2	736.6	1624.7
Grade of precipitate, per cent	72 4	90.9	62.8	76.7
Copper production, tons	93.3	690.1	463.6	1247.0
Average recovery, per cent. (1918)	81 8	98 5	64 8	84.9

TABLE 29.—*Cost of Precipitate Copper*, 1916 *to* 1918

	Cost per Pound Copper Bullion Produced
Labor	$0.01258
Supplies	0.00117
Repairs to flumes	0.00096
Freight to smelter	0 00021
Treatment at smelter	0 01925
Total	$0.03417

At the present time, all of the copper water is delivered from the upper levels to a concrete-lined reservoir on the 1000-ft. level through a series of diamond-drill drainage holes, $2^{13}/_{16}$ in. in diameter. From this reservoir, which has the usual settling baffles, the copper water is delivered to the 1000-ft. level flumes through 8000 ft. of 8-in. copper wound, redwood pipe. The pipe is laid in the 1000-ft. level Hopewell tunnel above the regular drainage ditch, on an average grade of 0.5 per cent.

Since this installation was completed, the average operating results from the 1000-ft. level plant, which now treats all of the copper-bearing water undiluted by barren solutions, have been as follows: Average assay of water at intake, 51.3 gr. per gal., average assay of water at discharge, 1.2 gr. per gal., average recovery, 97.6 per cent.; average grade of the precipitate, 87.6 per cent. A comparison of these figures with the results obtained from the combined plants, as given above, shows a net gain of 11.4 per cent. in grade of precipitate and 12.4 per cent. increase in recovery. This saving emphasizes the importance of delivering a clear undiluted copper solution to the precipitation plant.

Hopewell-tunnel Transportation

All ore hoisted through No. 5 shaft is dumped into storage bins above the 1000-ft. level. All ore broken in stopes above the 1000-ft. level is dumped into transfer raises which connect with these storage bins.

The ore from the storage bins is loaded into standard-gage bottom-dump cars of 220 cu. ft. nominal capacity, but of 280 cu. ft. capacity as loaded. This is equivalent to 20 tons of massive sulfide ore. Trains of fourteen cars are hauled an average distance of 8900 ft. to the Hopewell crushing plant by 25-ton locomotives. These locomotives are of Baldwin-Westinghouse bar-steel frame construction, equipped with two 75-hp. commutating-pole type motors, and automatic air brakes. The drawbar pull is 12,500 lb. at 7.1 mi. per hr. The average gross weight of the fourteen-car loaded train handled by each locomotive is 425 tons.

The cost of Hopewell tunnel transportation for the year 1918 was as here given. This cost covers the cost of loading, which increases rapidly whenever the ore from the upper levels becomes wet from surface drain-

	Per Ton Hauled
Labor	$0.054
Supplies	0.001
Power	0.005
Repairs	0.035
Total	$0.095
Credit limestone haulage	0.005
Net cost	$0.090
Cost per ton-mile	0 053

age. The cost of repairs includes charges for equipping the tunnel cars with air-brakes.

Mine Superintendence and General Occupations

The charge for superintendence includes the following: One mine superintendent, one assistant mine superintendent, one shaft construction foreman, three mine foremen, nineteen shift bosses. The average number of men employed under each shift boss is fifty-five. This high average is possible on account of the large stopes and the general concentration of mine workings.

General occupations includes the following departments: Engineering department, geological department, time office, samplers, underground watchmen (chiefly for fire protection), surface watchmen, change-room watchmen and janitors.

The total labor charge against the above account in 1918 was 24.3 cents per ton of ore delivered to the smelter.

General Repairs

The following cost statement for general repairs for the year 1918 covers all replacements and equipment necessary for enlarged under-

	Cost per Ton Ore Delivered to Smelter
Underground ore bins	$0 012
Mine track	0.037
Mine cars	0 038
Shaft stations and skip pockets	0 007
Drifts, crosscuts, and raises	0.049
Miscellaneous tools	0.008
Lighting system	0.014
Water and air lines	0 061
Surface fire protection	0 004
Air hoists	0 006
Mine model	0.002
Miscellaneous	0.004
Total	$0.242

ground operations, in addition to heavy expenditures for keeping all mine levels in repair.

General Expense

The total cost per ton for general expense for the year 1918 was distributed as follows:

	Cost per Ton Ore Delivered to Smelter
Mine inspection and first aid	$0.024
Water supply... .	0.029
Mine-office expense..... .	0.013
Engineering-office expense	0.004
Change house..... ...	0.014
Assaying.. ...	0.014
Bucking room .	0 005
Mine lights. .	0.010
Stable.	0.009
Proportion of administrative expense	0.079
Miscellaneous... . . .	0 018
Total .	$0 219

Ore Freight

The charge for transporting ore from the Hopewell crushing plant to the smelter at Clarkdale, a distance of 6.7 mi. over the standard-gage Verde Tunnel & Smelter R.R., is borne by the mine department. This cost, for 1918, was 23 cents per ton carried, equivalent to 3.4 cents per ton-mile.

Miscellaneous Charges

In addition to the operating charges previously noted, the following accounts are properly charged direct to mine costs: taxes, injuries and damages, fire insurance, depreciation of mine plant, preliminary steam-shovel charges. The total of these accounts for the year 1918 was $1.243 per ton of ore delivered to the smelter.

Recapitulation of Mine Costs

The following table is a recapitulation of mine costs for the year 1918, when 861,250 dry tons of ore were delivered to the smelter.

	Cost per Ton Ore Delivered to Smelter	Per Cent. of Total Mine Costs
Stoping	$2.070	39.15
Exploration and development	0 616	11.67
Diamond drilling	0.054	1.02
Hoisting	0 265	5.01
Drainage	0.025	0.46
Mine fires	0 004	0.08
Ventilation	0.031	0 58
Precipitating plants	0 014	0 26
Tunnel transportation	0 090	1.69
Superintendence and general occupations	0.243	4.59
General repairs	0.242	4.58
General expense	0.219	4.14
Ore freight to Clarkdale	0.230	4 35
Total operating cost	$4 103	77 60
Miscellaneous charges	1 184	22 40
Total mine costs	$5 287	100 00

From this summary, it will be noted that stoping and exploration and development costs constitute approximately 50 per cent. of the total mine costs.

DISCUSSION

Chas. A. Mitke,* Bisbee, Ariz. (written discussion).—During the past eight years, I have made frequent visits to the United Verde mine, and have watched the various changes and improvements developed in mining methods. One of the most important changes has been the gradual reduction in the variety of steel used and the final adoption of the Leyner and the quarter octagon. The Leyner steel is hollow round, 1¼ in. (3 cm.) in diameter, and is used for all drifting and stoping where very hard ground is encountered; the hollow quarter octagon, ⅞ in. in diameter, is used for both the water stopers and small jackhammer type of machine. Other important developments are the use of $\frac{1}{16}$ in. (1.58 mm.) changes and the reduction in the diameter of the drill bit.

At the United Verde mine, small machines and small steel have been tried in hard ground but the large machine and large steel have been found the more economical. In medium ground, however, the small machines and small steel are giving good satisfaction.

The substitution of wet for dry stopers in all hard rock mines is gradually being accomplished, the chief obstacle to an immediate change being the men, who generally prefer drilling in a dusty atmosphere to the slight discomfort from the water used in the drills. In one case, the miner inserted a wooden plug in the water hose and then gave as an

* Mining Engineer, Phelps Dodge Corpn.

explanation for drilling dry the statement that the entire water supply in the mine was shut off. Frequently, the men have simply turned off the water and drilled the holes dry.

The tendency at the present time is to mine lower-grade ores with correspondingly larger tonnages, both for direct smelting and for milling ore, and the problem is to obtain enough efficient labor. One of the solutions is the use of gravity methods in mining, wherever possible. Methods developed on the principle of incline raises and incline stopes allow practically all ore or waste to move by gravity, and also permit the production of much larger tonnages per man. It is on this principle that the Inspiration caving system has been developed. This system has incline raises at regular intervals, each one coming up under its particular area, or column, of ore. The ore is then drawn through the various incline raises into motor cars, being handled entirely by gravity. In a minor degree, this principle is the basis on which the incline stoping methods, such as the incline top slices in the Coronado mine of the Arizona Copper Co. and the incline cut and fills of the Junction mine of the Calumet & Arizona Mining Co., were developed. In both of these methods, the hand labor is reduced to a minimum and practically all ore and waste is handled by gravity.

Eight years ago practically all stopes in the United Verde were mined by the square-set system, the horizontal cut-and-fill being experimented with in a few stopes. Today, 58.5 per cent. of the ore is mined by the horizontal cut-and-fill, while only 8.5 per cent. of the ore is mined by the square-set. In the development of the horizontal cut-and-fill system, it soon became apparent that the walls surrounding the orebodies, in general, were very firm and that the ore stood very well with but little support. This made it possible to expand the cut-and-fill system by extending the stopes over large areas, in some cases over 100 ft. (30 m.) in width and several hundred feet in length. The only other examples in the Southwest that compare with this are the open horizontal cut-and-fill stopes in the Pilares of the Moctezuma Copper Co., at Nacozari, Sonora, Mexico, and the large incline cut and fills in the Junction mine of the Calumet & Arizona Mining Co., at Warren, Ariz. Even though the ore occurrence in these three mines is quite different, they have many similar characteristics. During recent years, experiments have been carried out with the incline cut-and-fill system in the Pilares mine. Wherever this system has been tried, it has proved to be well adapted to this character of ground, especially where the ore does not require too much sorting.

At present, only 1.4 per cent. of the ore in the United Verde mine is being stoped by the incline cut-and-fill system but experimental work may disclose a much wider use for the system.

Standardizing by North Butte Mining Co.

By Robert Linton,* New York, N. Y.

(Lake Superior Meeting, August, 1920)

This paper deals with work that has been carried on for over three years by the management and staff of the North Butte Mining Co. in an effort to standardize mining methods, to eliminate lost motion in the operating organization, and to improve the efficiency of the individual workman. Naturally, much of this work is an adaptation of ideas suggested by other mining operations or by practice in manufacturing and other productive industries. The studies, however, have developed some features that it is believed are new. Both new and adapted features are given in detail.

The normal capacity of the North Butte mines is about 2000 tons per day. Hoisting is done principally through the Granite Mountain shaft, which is 3740 ft. (1140 m.) deep and equipped with an electric hoist capable of hoisting 200 tons per hour from a depth of 4000 ft. (1219 m.). The lowest stoping level is the 3400-ft. The Granite Mountain shaft is connected with the Speculator shaft, about 850 ft. (259 m.) south, and with the Gem, about 1150 ft. (350 m.) east, which are 3000 (914.4 m.) and 2000 ft. (609.6 m.) deep, respectively. These two shafts are equipped with reversible fans for ventilation. The Speculator has a steam hoist with a capacity of 150 tons per hour from the 3000-ft. level. The Granite Mountain shaft is concrete lined from the 1600-ft. to below the 3000-ft. level and gunited with cement from the 1600-ft. level to the surface. Sections of the Speculator shaft with a total combined length of 500 ft. (152.4 m.) are also concreted.

There are approximately 36 miles (58 km.) of drifts and crosscuts in the mine, in which there are 15.2 miles (24.5 km.) of track. At normal capacity, about 75 stoping faces, 20 raises, and 30 sill development faces are working. Stoping is usually by the standard square-set method. Where the size of the oreshoot and character of the walls permit, the rill (or cut-and-fill) system is employed.

Scope and Limitations of Standardization

Standardizing mining operations presents problems quite different from the problems of the manufacturer. Methods of a successful

* President, North Butte Mining Co.

factory may be duplicated by copying its arrangement and equipment and following its production details; standardization can readily be applied to most manufacturing processes, especially if the article manufactured is produced in large quantities. Mining operations, on the other hand, are carried on under conditions that vary widely in different mines and even in the same mine. No two mines are alike and methods that prove efficient and economical in one mine are inefficient and uneconomical in another, where the formation and ore occurrence are of a different type. Standards to be effective must be specially worked out to meet the conditions of the particular mine in which they are to be employed.

It should be emphasized at the outset that any studies of standardization and efficiency should primarily be directed to helping the underground workman do his work more skilfully with less effort and less fatigue. It is an axiom in every line of industry that the most skilful workman is, as a rule, the workman who does his work with the greatest ease. Therefore, in making these studies the purpose has been to devise means for further development of skill, to make it possible for more men to possess it, to eliminate all avoidable delays and lost motion in the working organization, and thus enable men to do the maximum amount of work without injurious overexertion. We believe that, given good working conditions and a square deal, the average workman would rather feel that his day's work turns out something of benefit to his employer than not, and is willing to do his best if he knows his efforts are recognized and compensated for on a fair basis.

The miner was formerly a member of a highly specialized trade—skilled in the manipulation of drills and the use of explosives, familiar with varying kinds of rock, trained to observe anything and everything that affected the particular mining job on which he was employed. There are still many such skilled miners working underground and every mine superintendent is on the lookout for them, but there are also many who do not possess this skill and who work much harder without accomplishing as much because their efforts are not utilized to the best advantage. Standardization of methods benefits such men particularly.

The most important factor in any industrial operation is not the method but the man. In carrying out the work described in this paper, the great fact has been kept in the foreground that men are human beings and not cogs in a machine. N. B. Braly, general manager of the company, devoted a great deal of time and personal study to the problems involved, bringing to them his intimate knowledge of mining practice. The writer feels that Mr. Braly has made a notable contribution to the industry in all this work and especially in several novel features devised by him, which will be described later. He was ably assisted by other members of the staff, among whom may be mentioned L. D. Frink,

superintendent, E. H. Hewitt, assistant superintendent, and E. F. Raney, efficiency engineer.

Equipment

Air Drills

The selection of machine drills involved an investigation of the following points: steady operation, fast drilling speed, convenience in handling, low air consumption. It is unnecessary to say that every experienced mine superintendent selects his drills for their practical rather than theoretical efficiency. The drill that stays underground and makes the fewest trips to the repair shop is much more desirable and economical than the drill that shows a low air consumption but is not rugged enough to stand the hard usage of underground work. Similarly, while fast drilling speed is important, steady operation is even more important. Time studies have shown that only from 14 to 36 per cent. of the total time of the machine operator on development work, depending on the ground, is actually consumed in drilling; the rest of the time is taken in loosening and tightening bolts, changing steel, etc. The complete record for a shift's work, drilling a round of holes in a timbered crosscut with medium ground, is given in Table 1.

TABLE 1.—*Drilling Record of One Shift in Medium Ground*

No of Hole	Kind of Hole	Vertical Angle, Degrees	Horizontal Angle, Degrees	Actual Drilling Time		Depth, Inches	Feet per Minute
				Min.	Sec.		
1	Up	15	3	5	37	66	0.98
2	Down	15	7	9	21	62	0.55
3	Down	15	7	13	52	62	0 38
4	Down	10	17	8	34	67.5	0.65
5	Down	10	17	6	41	57	0.71
6	Horizontal		19	8	36	70	0.68
7	Horizontal		19	5	25	66	1 01
8	Up	15	3	4	24	66	1 25
9	Up	15	15	4	25	63	1.18
10	Up	15	15	4	38	68	1 22
11	Down	10	3	6	35	60	0 78
12	Down	10	20	9	39	71	0 61
13	Down	10	20	8	43	36	0 34

The average results were: upward sloping holes, 1.16 ft. per min.; downward sloping holes, 0.66 ft. per min. The work of drilling upward sloping holes was 75 8 per cent. more efficient than the drilling of downward sloping holes. These averages do not include holes 3 and 13, which are not representative, as the steel stuck on striking ore.

The results of these studies indicate that there is room for betterment in drilling equipment by improving the accessories that go with the machines used for drifting and crosscutting, such as the arms, clamps, etc. Even if the drilling speed of a machine should be increased 100 per cent., the day's work, under present conditions, in average rock would not be shortened over 15 per cent. Any improvement that would save 1 hr. of the time now lost would equal a 10 per cent. increase in drilling speed.

Another opportunity for improving drilling equipment would be the development of a successful wet stoper. The present standard stoping drills are operated dry and, consequently, make large volumes of dust in drilling dry faces. Dust has been eliminated in drifts and crosscuts by using the wet type of mounted drill; but while the studies included experiments with every wet type of stoping drill that has been put on the market, none was found that was satisfactory to the drill operators. Both spray and wet-jet types have been used, but in comparison with the dry stopers, the drilling speeds of the wet stopers are lower, the repair costs are much higher, the time lost for repairs is much greater, and the operators object more strenuously to the mud and water from the wet stopers than they do to the dust from the dry stopers. The development of a wet stoper that would apply the water so as to keep down the dust without soaking the men would be a most important improvement and would contribute greatly to the health and comfort of miners working in stopes and raises.

The drills used are the No. 448 Ingersoll-Leyner, the Waugh Turbro; and the D X 61 Sullivan for crosscutting and drifting; the 16 V Waugh and Ingersoll-Rand B C 21 stoping drills for raising and stoping. These drills are not necessarily better than any other types or makes but they have been found to be those best adapted to North Butte conditions. In other operations with which the writer is connected, an entirely different selection of drills has been made by following the same policy of selecting the best drill for the particular work in hand.

In the drill-repair shop, there are swinging leaves with illustrations of the various parts of the machines and giving the cost of each. There is also a Rand file with a swinging leaf for each type of drill, and a separate card for every drill in use. On this card are entered the material used and work done each time a drill is brought to the shop for repairs. From this record, total repairs for any period can be figured; and from the underground records, the repair costs per 100 ft. of ground drilled can be obtained for any type of drill.

Drill Steel

The determination of standards for drill steel was governed by similar considerations. The cost of handling the steel from the mine to the shop, sharpening it, and returning it to the working face is a large factor

in the expense of drilling. The total feet drilled per sharpening is, therefore, of as much importance as the drilling speed. Careful tests with various steels proved conclusively that 7/8-in. (22.23-mm.) so-called quarter octagon steel (square steel with truncated corners) best fulfilled the requirements for raising and stoping. As it is expected that a successful wet stoper will eventually be made, the standard steel should be hollow and the old standard cruciform steel is a solid steel. In early experiments with wet stopers, octagon steel was used but the mud running down the steel into the chuck occasioned excessive wear; by using quarter octagon steel the wear has been greatly reduced.

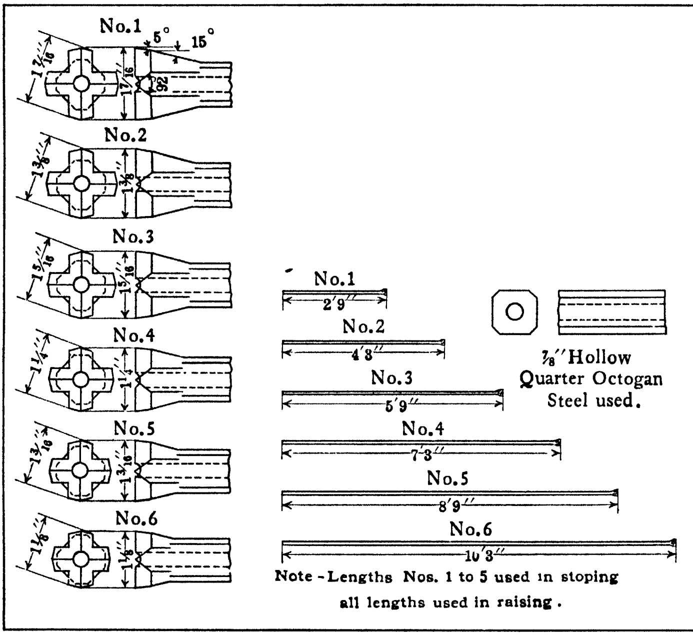


FIG. 1.—DRILL STEEL FOR STOPES AND RAISES, SHOWING DRILL LENGTHS AND BITS USED.

Comparative tests were made with both 1-in. and 7/8-in. quarter octagon and the latter was adopted as the standard. Naturally the 7/8-in. steel permits the use of smaller bits with consequent higher drilling speed, less wear and tear on the machine, and materially lower powder consumption since 7/8-in. powder is used while with 1-in. quarter octagon 1 1/4-in. powder is employed.

Speed tests showed a 50 per cent. increase in drilling speed, and in some cases 100 per cent., over the speed attained when using the 1 1/8-in. cruciform steel. Details of one set of tests are given here. In this

case the average of all tests showed 67 per cent. higher speed when using the ⅞-in. steel.

TEST No. 1

Old Standard.—1⅛ in. cruciform steel; bits, 2 in., 1⅞ in., 1¾ in., 1⅝ in., 1½ in.

	Total Time Min.	Total Time Sec.	Time Out Min.	Time Out Sec.	Actual Drilling Time Min.	Actual Drilling Time Sec.	Depth in Inches	Inches per Minute
Hole No. 1	30	5	16	8	13	57	95	6.80
Hole No. 2	17	10	5	51	11	19	98	8.65
Total	47	15	21	59	25	16	193	7.65

New Standard.—⅞-in. quarter octagon steel; bits, 1⅜ in., 1⁵⁄₁₆ in., 1¼ in., 1³⁄₁₆ in., 1⅛ in.

	Total Time Min.	Total Time Sec.	Time Out Min.	Time Out Sec.	Actual Drilling Time Min.	Actual Drilling Time Sec.	Depth in Inches	Inches per Minute
Hole No. 1	30	10	17	13	12	57	96	7.40
Hole No. 2	14	38	8	18	6	20	93	14.70
Hole No. 3	9	50	4	40	5	10	82½	16.00
Total	54	38	30	11	24	27	271½	11.18

TEST No. 2

Old Standard.—1⅛-in. cruciform steel; bits, 2 in., 1⅞ in., 1¾ in., 1⅝ in., 1½ in.

	Total Time Min.	Total Time Sec.	Time Out Min.	Time Out Sec.	Actual Drilling Time Min.	Actual Drilling Time Sec	Depth in Inches	Inches per Minute
Hole No. 1	17	15	9	48	7	27	98	13.15
Hole No. 2	16	57	3	33	13	24	82	6.12
Hole No. 3	21	9	10	16	10	53	88	8.10
Hole No. 4	19	20	9	35	9	45	90	9.23
Hole No. 5	18	37	9	56	8	41	98	11.30
Hole No. 6	18	47	7	32	11	15	93	8.25
Total	112	5	50	40	61	25	549	8.95

New Standard.—⅞-in. quarter octagon steel; bits 1⅜ in., 1⁵⁄₁₆ in., 1¼ in., 1³⁄₁₆ in., 1⅛ in.

	Total Time Min.	Total Time Sec.	Time Out Min.	Time Out Sec.	Actual Drilling Time Min.	Actual Drilling Time Sec	Depth in Inches	Inches per Minute
Hole No. 1	20	40	19	0	1	40	42	25.2
Hole No. 2	8	55	4	26	4	29	91	20.3
Hole No. 3	11	35	7	24	4	11	92	22.0
Hole No. 4	10	25	3	52	6	33	92	14.06
Hole No. 5	19	53	13	51	6	2	91	15 1
Total	71	28	48	33	22	55	408	17.81

COMPARISON OF RESULTS

	No of Holes	Average Net Drilling Time Min.	Average Net Drilling Time Sec.	Average Depth of Hole, In.	Average Feet Drilled per Minute
Test No. 1 1⅛-in. steel	2	12	38	96.5	7.65
Test No. 1 ⅞-in. steel	3	8	9	90.5	11.18
Test No. 2 1⅛-in. steel	6	10	14	91.5	8.95
Test No. 2 ⅞-in. steel	5	4	35	81.6	17.81
Total 1⅛-in. steel	8	10	50	92.75	8.57
Total ⅞-in. steel	8	5	54	84.93	14.33

For drifts and crosscuts 1¼-in. hollow round steel has been found most advantageous and has therefore been adopted as the standard. It is

ample in size, convenient to handle, and being made up with lugs on the shank is easily extracted from the hole by merely cranking back the machine under half head of air. The advantage of quick extraction is apparent since actual drilling time is seldom 25 per cent. of the shift.

The cross-bit was adopted as the standard for all drills, whether of ⅞-in. or 1¼-in. steel. It was demonstrated that this style is well adapted to all classes of ground at North Butte, is easy to keep to exact gage in sharpening, and gives a high average of feet drilled between sharpenings. Some other types of bit will drill somewhat faster, but the cross-bit has sufficient speed and combines speed with other requirements in a more satisfactory manner than any other type tested.

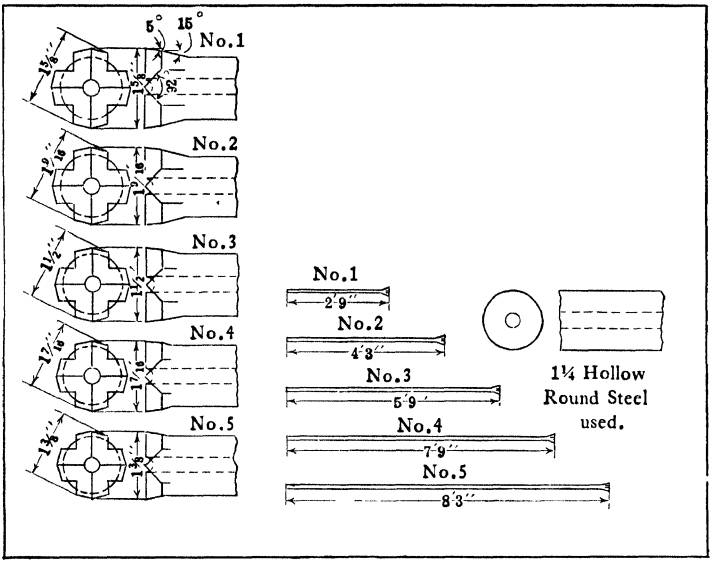


FIG. 2.—DRILL STEEL FOR DRIFTS AND CROSSCUTS, SHOWING DRILL LENGTHS AND BITS USED.

Fig. 1 shows the standard set of drill steel for stopes and raises and Fig. 2, the set for drifts and crosscuts. It will be noted that the cutting and reaming edges are made long. It is of the utmost importance to have the 5° slope near the point of the steel, as shown. This puts what is called a reaming edge on the steel and makes the wear at this point slower than where the sharper angle of 15° is used, which was the old standard. The bits are forged to a clearance of 1/16 in (1.59 mm.) although the actual average wear shown by tests is about 1/32 in. Standardizing drills to permit changing on 1/16-in. reduction of diameter has been found to have two important advantages: First, a material increase

in drilling speed is obtained; second, bits of average smaller diameter are used [the starting bit is $\frac{5}{8}$ in. (15.88 mm.) smaller], which reduces wear on the machine from percussion and consequently cuts down repairs.

Auxiliary Drill Equipment

The standard equipment for stoping is as follows:

1 stoping drill
1 hose for stoper
hose gaskets
3 sets drill steel ($2\frac{1}{2}$ to 9 ft.)
1 jackhamer
1 set jackhamer drill steel
1 steel wrench
1 monkey wrench
1 oil bottle
1 round-point shovel
1 square-point shovel
2 8-lb. hammers
1 ax
1 saw
1 pick
1 pinch bar
1 tamping stick
1 powder sack
4 platform hooks
1 Bacon timber hoist with rope
1 weight for timber hoist
1 pulley for hoist rope
1 wheelbarrow
1 water keg

The standard equipment for raising is as follows:

2 stoping drills
2 hose for stopers
hose gaskets
3 sets drill steel ($2\frac{1}{2}$ to 9 ft.)
1 oil bottle
1 short pinch bar
1 long pinch bar
1 14-in. pipe wrench
1 monkey wrench
2 steel wrenches
2 picks
2 axes
2 8-lb. hammers
1 square-point shovel
1 round-point shovel
1 powder sack
2 tamping sticks
8 platform hooks
1 water keg
1 bell line rope
1 Bacon timber hoist with rope
1 weight for hoist rope
1 pulley for hoist rope
1 gin pole
2 saws

The standard equipment for drifting or crosscutting is as follows:

1 water Leyner type machine
1 column bar for machine
1 arm for column bar with machine clamp
3 sets drill steel for Leyner type machine
1 jackhamer
1 set jackhamer drill steel
2 air hose
1 water hose
1 drilling bench
1 monkey wrench
1 pipe wrench
1 chuck or clamp wrench
2 picks for cleaning steel
1 "Submarine" (water tank for drills)
1 scraper
1 pinch bar
1 tamping stick
1 blowpipe
1 oil bottle
4 platform hooks
4 timber dogs
1 saw
1 water keg
1 powder sack
1 level
1 grade stock
1 pick
1 ax
1 8-lb. hammer
1 round-point shovel
1 square-point shovel
mine cars

The supports of the standard drilling bench are hinged and fold up under the plank when the bench is not in use. The drill mountings are stamped with graduated arcs, reading to 5°. These graduations are coördinated to a line on the arm in such manner that when the zero of the arc is set at this line the drill stands in a horizontal position. The vertical angle required to be drilled, either up or down, is obtained by setting this angle against the line on the arm, when the drill is in proper position. Horizontal angles are obtained by reading a similar graduated arc on the saucer of the drilling machine. For this angle to be correct, the arm

of the bar must be set at right angles to the direction of the crosscut. This is accomplished by sighting along a groove in the arm near the bar at some object back in the crosscut, which always puts the arm in the correct position.

To lay off drill holes in raises, a Lufkin jointed rule is used, having a level on one arc and a graduated arc showing the angle between the arms. With this the stoping drill can readily be set at any required angle.

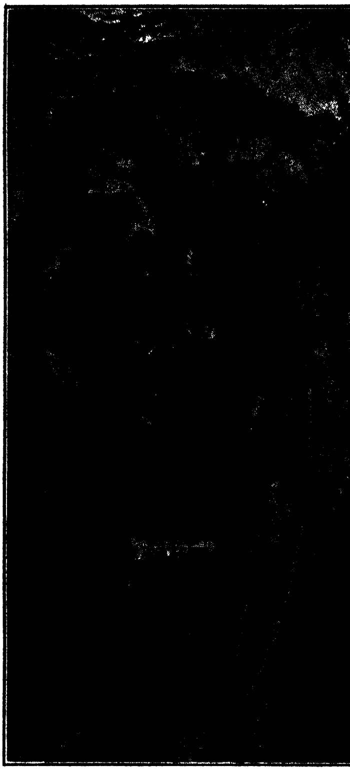


FIG. 3.—DRILLING BENCH SET UP.

FIG. 4.—DRILLING BENCH TAKEN DOWN.

Ventilating Equipment

The main ventilating currents through the mine are furnished by large Sirocco fans, on the surface, at the Speculator and Gem shafts. The currents are controlled by doors so as to afford the best possible distribution of air through the workings of the mine. Many stopes, raises, and development faces are, however, not in the path of the circulating air. To ventilate these, portable motor-driven Sirocco fans connected to

canvas tubing are in use. The application of canvas tubing to ventilating was made feasible by the flexible coupling and method of suspension invented by Mr. Braly.

Canvas tubing possesses the following important advantages over metal pipes:

1. It is much more easily and quickly installed, and when not carrying air hangs collapsed at the side of the working where it takes up very little room.

2. Since it hangs collapsed during the blasting of the round and is expanded by the air pressure of the fans turned on immediately after-

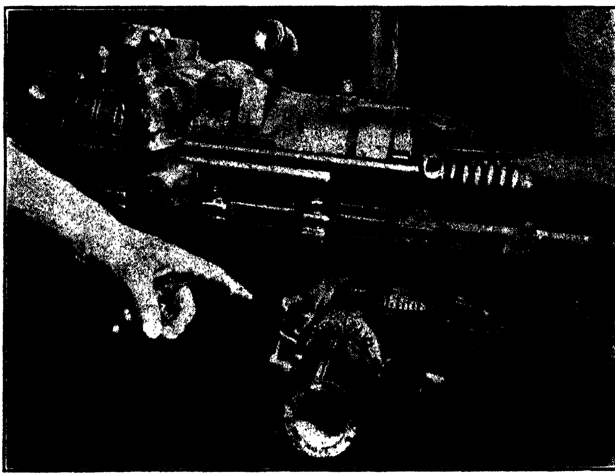


FIG. 5.—GRADUATED DRILL MOUNTINGS.

wards, ventilation can be carried close to the face. In ordinary crosscuts, it is impossible to keep metal pipe closer than 100 ft. (30.5 m.) to the breast because the concussion in blasting tears it down; even then, unless the end is tightly covered with a board, the pipe may collapse for a length of 60 to 70 ft. (18 to 21 m.).

3. Falls of rock in a drift ruin metal pipe permanently, whereas canvas pipe frequently is not even punctured; or if damaged it can easily and quickly be repaired.

4. Canvas tubing can, without the slightest difficulty, be run through stopes, over and around timber, and in numerous places where the fitting of metal pipe would be too slow and expensive to be feasible.

5. The miners can put on lengths of canvas tubing and deflect the air so as to blow directly upon them, thus removing the layer of warm air surrounding their bodies. This is of great importance, especially on the lower levels where rock temperatures are high.

Of course the friction in canvas tubing is greater than in an absolutely tight metal pipe of the same diameter; but in ordinary practice the reduction of volume and pressure due to the higher friction in the canvas tubing is considerably less than the reduction due to leaks in metal pipe. It is difficult to get any satisfactory ventilation from an ordinary metal pipe 1000 ft. (305 m.) long, whereas excellent ventilation has been obtained from canvas tubing over 2400 ft. long. The size of tubing and the corresponding size of fan are determined by the size of the workings to be ventilated and the distance the air must be carried. Standards based on clearing a working face within 35 min. after blasting are shown in Fig. 6. The equipment can be used to ventilate at much greater distance by consuming correspondingly longer time.

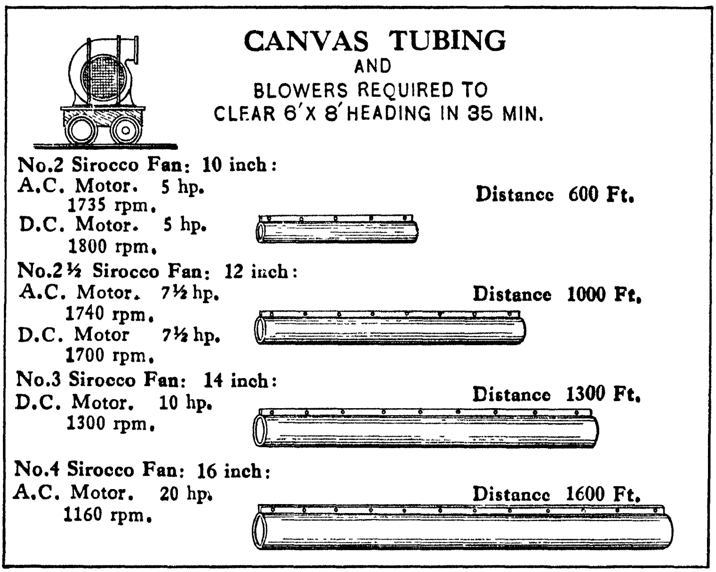


FIG. 6.—STANDARDS FOR CLEARING A WORKING FACE WITHIN 35 MINUTES.

Both direct- and alternating-current motors are used for driving the fans; direct current is preferable since the motors can be attached direct to trolley lines anywhere in a few minutes' time. The best practice is to use a larger motor than the rating of the fan specifies, in order to allow a margin of safety for overheating where necessary to use it under temperature conditions above normal. Fans No. 2, 2½ and 3 are self-contained, being mounted on trucks complete with starting apparatus, ready to connect to the transmission wire. The dimension mountings permit lowering the mounted fans on the cages without taking apart The No 4 Sirocco fan is too large to lower on the mine cage without

taking apart; hence, it is not mounted on a truck, but where used is placed on a concrete foundation. Practically all the fans used are the smaller sizes since most of the ventilating distances are comparatively short

FIG. 7.—MOUNTED SIROCCO FAN.

FIG. 8a. FIG. 8b.
FIG. 8a AND 8b CANVAS TUBING IN USE.

Ore Handling

The standard chute and standard gate are shown in Figs. 9 and 10. The two wooden doors control most conveniently the flow of rock, but an iron door is also placed at the mouth of the chute to facilitate shutting it off at any time without having runs of rock.

Mine Cars

Mine cars are of the ordinary end-dumping type, holding 0.675 ton. They are handled in trains of twelve cars by Westinghouse-Baldwin trolley locomotives. One advantage in the use of trolley locomotives for haulage is that current is available all over the mine for ventilating

STANDARD GIRT CHUTE

INSTRUCTIONS: In starting to put in a chute, see that the ends of the bottom plank are directly over the near rail and 5 ft. 3 in. above it. The center of the standard should line with the center of the post at the top and be exactly plumb, regardless of the batter on the sill posts. Bottom plank to be on a 45° slope; chute jaws 2 to 6 in. above ends of chute plank. In heavy ground make end of chute 5 ft. 8 in. above rail.

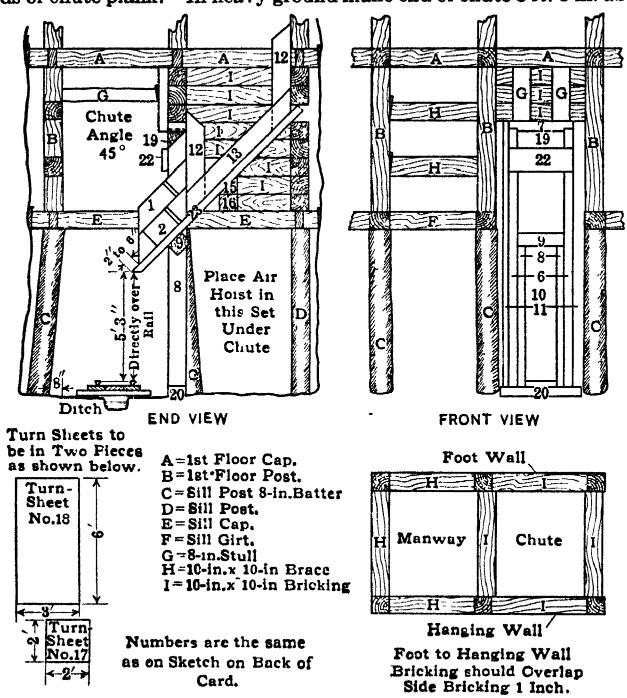


FIG. 9.—FACE OF CARD SHOWN IN FIG. 10.

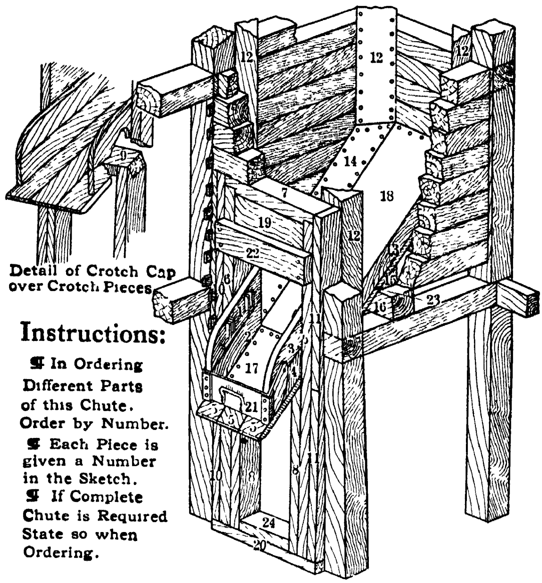
fan motors. Tracks are 18-in. gage (46-cm.) with 25-lb. (11.34-kg.) rail as standard all over the mine, both on main and auxiliary haulage ways, with 0.4 per cent. grade in favor of the load. Fig. 11 shows the standard switch in use. Oil is kept in small sunken tanks fitted with pumps.

Mechanical Loading and Protecting Raises

Considerable experimenting has been done with a view to finding or developing a satisfactory machine for loading broken rock at the faces

of drifts and crosscuts. The Hoar loading shovel was used for a time in one crosscut, but the size of the crosscut (6 by 8 ft.) was too cramped for its efficient operation. At the time of writing this paper, a belt-conveyor

STANDARD GIRT CHUTE



	Piece	No.	Dimension		Piece	No	Dimension
1	Left chute jaw	1	3″×10″× 3′10″	13	Right cor. bottom pc.	1	10″×10″×6′8″
2	Left chute jaw	1	3″×12″× 3′10″	14	Left cor. bottom pc	1	10″×10″×6′8″
3	Right chute jaw	1	3″×10″× 3′10″	15	Bottom brace	1	10″×10″×3′6″
4	Right chute jaw	1	3″×12″× 3′10″	16	Bottom brace	1	10″×10″×5′10″
5	Bottom plank	3	4″×10″×10′10″	17	Bottom turn sheet	1	2′×2′
6	Standards	2	5″×10″×12′2″	18	Bottom turn sheet	1	6′×3′
7	Standard cap	1	5″×10″×3′	19	Filling piece*	1	10″×10″×2′2″
8	Crotch pieces	2	4″×10″× 6′6″	20	Sill piece	1	4″×10″×3′
9	Crotch cap	1	6″×10″× 2′2″	21	Chute door	1	
10	Left filling piece	2	4″×10″× 6′	22	Front piece	1	4″×10″×3′
11	Right filling piece	2	4″×10″× 6′	23	Bottom pieces	2	4″× 7′ ×7′8″
12	Corner vertical p	4	10″×10″×5′—45%	24	Spreader	1	3″×10″×1′ 6″

* Don't put in filling piece until turn sheets have been placed!!

FIG. 10—BACK OF CARD SHOWN IN FIG. 9.

type of loader is being tried, but its use has not proceeded far enough to demonstrate whether it can be used satisfactorily.

The tops of raises were formerly protected by transverse guard timbers

4 by 12 in. (10.16 × 30.48 cm.) spaced 12-in. in the clear. Hinged latticed doors fabricated from 1½ by ¼-in. (3.81 × 0.63 cm.) iron are now used. These doors permit a freer flow of air through the raises and afford better protection against accidents.

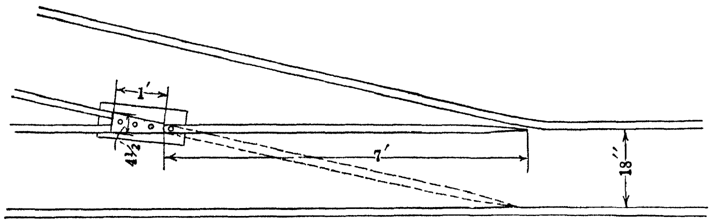


FIG. 11.—STANDARD SWITCH POINT AND FROG.

METHODS

Standard Drill Rounds

Probably the most important series of studies made at North Butte were those that fixed standard drilling rounds for both development and stoping faces. The objects of this work were: To reduce the amount of powder used, to increase the tons of rock broken per foot of drill hole, to increase the average footage broken per round, to save time usually taken by men in deciding just where holes should be placed.

Theoretical Considerations

The blasting formula stated by Henry S. Drinker (following Schoen) in his work on tunneling, published in 1878, has been assumed by most authorities to be approximately correct. This formula is as follow:

W = weight, in ounces, of blasting agent of a given kind.
L = line of least resistance, in feet; this is the shortest distance from the center of gravity of the charge of blasting agent to the nearest free face of rock.
C = charging coefficient, or coefficient of rock resistance.

Then,

$$W = CL^3$$

Transposing this formula the coefficients at North Butte were determined as follows:

Soft ground, 0.015; medium ground, 0.024; hard ground, 0.043; extra hard ground, 0.1. However, the results did not check with this formula when it was applied to determining the weight of powder necessary to blast holes under varying conditions of depth and angle. It has been used in connection with blasting with black powder, but with the

high-power dynamites now employed, the velocity of the explosion, as well as the force of the blow, is much more intense. The tests at North Butte seem to demonstrate that the weight of the charge is proportional to the square, and not the cube, of the line of least resistance, so the formula becomes

$$W = CL^2$$

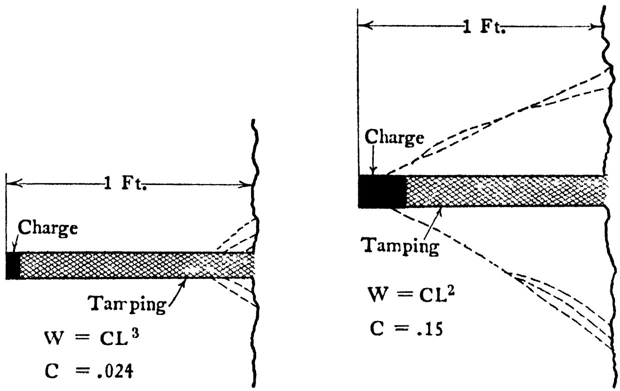


FIG. 12.—EXPERIMENTS COMPARING RESULTS OF BLASTING ACCORDING TO FORMULAS $W = CL^3$ AND $W = CL^2$

A comparison of the results of blasting according to each formula (using 60 per cent. dynamite) is shown in Fig. 12.

Fig. 13 shows the theoretical weight of explosives required to blast holes of varying depth, as figured by the original and modified formulas. For a hole 1 ft. deep, the original formula calls for only one-sixth as much explosive as the modified. Tests have shown that the larger quantity is more nearly correct for high explosives.

Rock coefficients based on the formula $C = \frac{W}{L^2}$ were found to be as follows:

	DRIFTS	STOPES 8-FT. ROUND	STOPES 4½-FT. ROUND
Soft ground	0 08	0.05	0 08
Medium ground	0 15	0 12	0.12
Sharp ground	0 18	0 15	0.18
Hard ground	0.3	0 36	0 3
Extra hard ground	0 6	0 6	0.5

Mr. Braly advances the following theoretical conclusions suggested by the blasting tests made. In these conclusions, the line of least resistance is taken as the shortest distance from the center of gravity of the charge of blasting agent to the nearest free face of rock:

1. The amount of powder is roughly in proportion to the surface to be sheared.

2. In a cut hole this is a cone, with the line of least resistance as the axis, and the amount of powder is in proportion to the square of the line of least resistance in feet; in other holes, the shearing surface is only a section of a cone.

3. The softer the ground, the wider is the apex angle of the cone.

4. After the shearing has taken place, the subsequent action of the powder is shattering the rock and blowing it out of the hole.

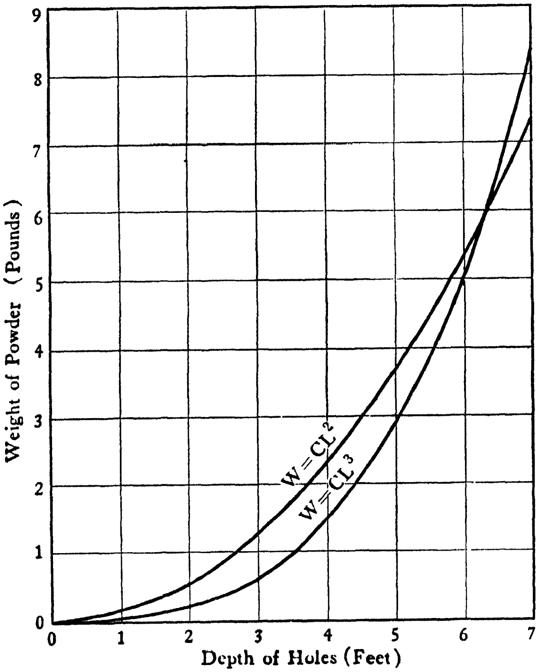


FIG. 13.—THEORETICAL WEIGHT OF EXPLOSIVE REQUIRED TO BLAST HOLES OF VARYING DEPTH FIGURED ACCORDING TO FORMULA $W = CL^2$ AND $W = CL^3$.

5. High-grade powder gives more of its energy for shattering and less for heaving.

6. Low-grade powder gives more of its energy for heaving and less for shattering.

7. With a slow powder, where heaving is desired, the amount will probably vary more nearly as the cube of the depth; but with high explosives with shattering effect it will vary more nearly as the square of the depth.

The tests indicate an increase in intensity of the detonating wave as it progresses toward the bottom of the hole (Fig. 14, after Grant H. Tod). Naturally, with greater depth of hole and longer line of least resistance,

greater explosive intensity is required. The force of the explosion should break the rock to as near the bottom of the hole as possible. However, the entire charge should be exploded at as nearly the same time as possible, which is accomplished by placing the primer just outside the middle of the charge.

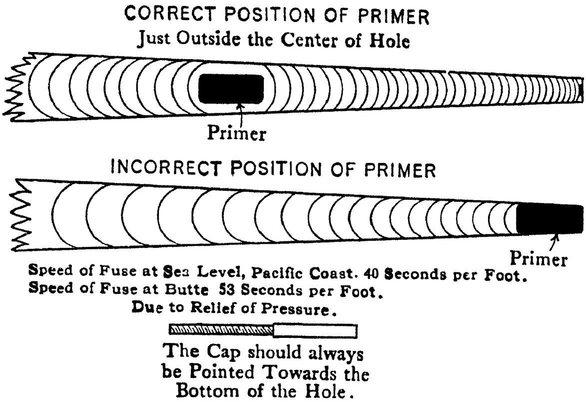


FIG. 14.—CHART SHOWING INTENSITY OF DETONATING WAVE IN BLASTING.

Practical Considerations

Probably the most conspicuous fact brought out by the drill-round studies was that few drill operators or shift bosses are able to form a definite picture of the plane in which the drill holes bottom. Obviously blasting starts from this plane, and its effectiveness depends on the location of the bottoms of the holes with reference to one another and the angles they make with the plane.

When investigating this, a method[1] used at the University of California was employed. Two frames corresponding in size and shape to a standard crosscut were built, and covered with woven wire. These were set up 5 ft. 6 in. (1.68 m.) apart, corresponding to a standard round to break 5 ft. Men were given the problem of setting up a drill and pointing holes for what they considered a good round. The location of the holes was marked on both screens, the back screen showing the position and angles of the holes in the blasting plane. In many cases, rounds were laid out that would not break clean to the bottom although frequently more holes were pointed than would be required with exact spacing. Other practical factors were ascertained by observation of drill operators and by blasting special rounds. The most important may be summarized as follows:

1. The principal cause for rounds breaking short or unevenly was incorrect spacing of holes in the blasting plane.

[1] Illustrations showing the use of these screens appeared in *Min. & Sci. Pr.*, Dec. 13, 1919.

2. More powder than actually required was used, as a rule, to insure blasting of rock masses lying between holes that happened to bottom too far apart.

3. The rule that high-grade powder shatters and low-grade powder heaves was not always observed and the explosives, therefore, were not always used to best advantage.

4. Since upward-sloping holes can be drilled much more rapidly than downward-sloping holes, the round should be drilled with as large a proportion of holes sloping upwards as possible. The average of the mine shows a gain of 45 per cent. in favor of upward pointing.

5. Incorrect placing of the primer may reduce the efficiency of the blasting as much as 40 per cent. The amount of carbon monoxide in

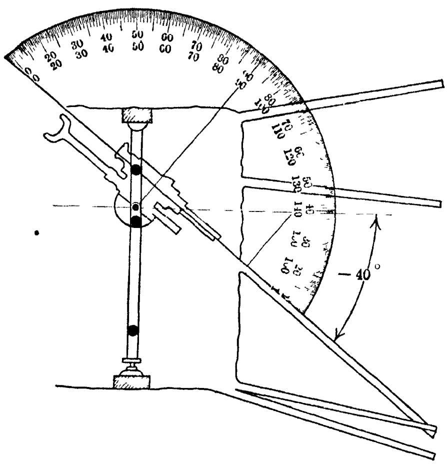


FIG. 15.—CELLULOID PROTRACTOR USED IN LAYING OUT DRILL ROUNDS.

the smoke after blasting is a good indicator; the less efficient the blasting, the more carbon monoxide is generated.

6. As the proportion of time consumed in actual drilling is small (varying from 14 to 35 per cent. of the total time of the operator) the drill round should be laid out so as to reduce to a minimum the necessity of moving bars, columns, arms, etc.

7. In crosscuts or drifts, where there is but one face to break to, the powder should be concentrated near the bottom of the hole. This is not the case in stopes where there are two faces to break to.

Laying Out the Drill Round

The standards were determined by drilling and blasting a large number of experimental rounds. Each round was carefully plotted to ½ in. (12.7 mm.) scale on paper in advance of drilling. The position and

angle of each hole in the blasting plane was determined and its line projected back to the position of the machine drill. The lines were then adjusted to converge at as few points as possible, and thus reduce the number of times necessary to change the position of the drill.

In plotting these proposed drill rounds, a template of a rock drill was cut out of celluloid to exact scale. A subsequent improvement was to combine the template with a 5-in. (12.7-cm.) celluloid protractor, orienting it so that the center of the column arm and the center of the protractor coincide, the bottom of the protractor being the center line of the

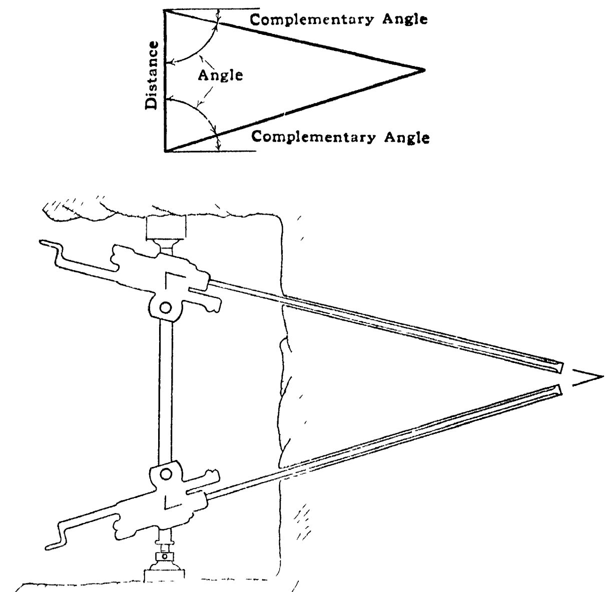


FIG. 16.—GEOMETRY APPLIED TO THE ROCK-DRILL ROUND.

drill steel. This drill protractor (shown in Fig. 15) is very convenient and facilitates rapid plotting of the rounds.

This method of laying out a drill round is a simple application of geometry to the problem. Given the position of the bottoms of the holes, the angles and height of the machine necessary to drill the holes to reach these positions can be accurately determined. There is no haphazard work and the effect of different positions and angles of holes can be analyzed intelligently. Figs. 16 and 17 illustrate this.

Standard Rounds for Drifts and Crosscuts

Fig. 18 shows the instruction card for standard round for a timbered drift or crosscut. Fig. 19 shows the plan of the drift giving the coördina-

tion of the drilled face to the timbering. From this it will be noted that 2 ft. 6 in. (0.76 m.) has been established as the standard distance of the column from the face. This distance is always observed in untimbered crosscuts, but in driving timbered drifts and crosscuts the column is set up 3 ft. ahead of or 1 ft. behind the last set of timber, as the case may be, the measurements being taken from outside of the cap. A 5 ft. 6 in. (1.68 m.) round is specified, but it is contemplated to pull full 5 ft., the extra 6 in. being added to make up for short rounds. If 5 ft. can be pulled regularly with each round, and it has been found that in most

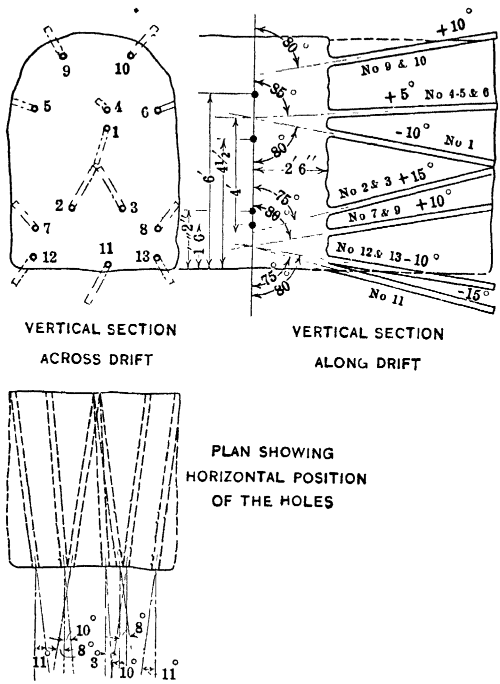


FIG. 17.—METHOD OF LAYING OUT ROCK-DRILL ROUND.

cases this can be done, this makes the best standard arrangement that can be devised for timbered drifts. The holes are laid out so that when the round is blasted, the face will be the widest part of the drift, allowing the timbers to be placed readily without plugging. Otherwise, it is frequently necessary to drill and shoot plug holes before the timber can be placed.

The instruction card is self explanatory, giving the position, vertical and horizontal angles, and depth of the holes, the amount and grade of powder to be used and order of blasting of round. It is unnecessary to

SERIES "C"

TIMBERED HEADING.

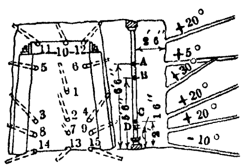
515 IN ORE

DATE...... .. ..EXPLOSIVES ISSUED
PLACE....... .. . ..BOSS'S NO.. . . ..BEFORE LUNCH
MAGAZINE NO... .. ...MINER'S NO........ .. ..AFTER LUNCH...... ... .

	Holes		Powder		Number of Primers	Sticks of Tamping	Separately Blasted Heading
	Number	Depth	35%	60%			
Round							
Plugs							
Total							

CHECK HARDNESS OF GROUND	MEDIUM GROUND	SOFT GROUND
ACROSS BOTH PORTIONS OF CARD	MEDIUM GROUND	SOFT GROUND

MEDIUM GROUND TO BREAK 5′6″

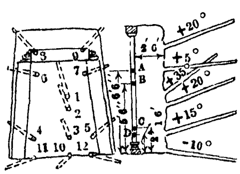


Holes	D	Swing	Holes	D	Swing
1–2–7–10–13	2″	3°	8– 9	14″	19°
3–4	14″	18°	11–12	14″	15°
5–6	14″	16°	14–15	14″	20°

Holes		Powder	
Number	Depth	35%	60%
1	5		8
2–3–4	6		8
5–6	6		7
7–8–9	5½–6		7
10–11–12	5½–6		6
13–14–15	5½–6	8	
Round	87½	24	85

Holes	Position		Angle	Steel
1	C	O	+30°	5½
2–3–4	C	U	+20°	6½
5–6	B	O	+ 5°	6½
7–8–9	D	U	+10°	6–6½
10–11–12	A	O	+20°	6–6½
13–14–15	D	U	–10°	6–6½

SOFT GROUND. TO BREAK 5′6″



Holes	D	Swing	Holes	D	Swing
1– 2	2″	3°	6– 7	14″	16°
3–10	2″	3°	8– 9	14″	15°
4– 5	14″	18°	11–12	14″	19°

Holes		Powder	
Number	Depth	35%	60%
1	5		7
2	6		7
3–4–5	5½–6	7	
6–7	6	6	
8–9	6	6	
10–11–12	5½–6	6	
Round	70	63	14

Holes	Position		Angle	Steel
1	C	O	+35°	5½
2	C	O	+20°	6½
3–4–5	D	U	+15°	6–6½
6–7	B	O	+ 5°	6½
8–9	A	O	+20°	6½
10–11–12	D	U	–10°	6–6½

FIG. 18.

go into the experimental details by which these standards were arrived at, but it may be noted that many different arrangements were tried and it is believed that the round illustrated represents as efficient practice for particular conditions as can be devised.

The standard round for drifts and crosscuts is drilled from only four heights of the arm. These positions are located by drilling holes in the column in which a spike may be inserted. When the arm rests on the spike, the drill is at the proper height. In this way the time required to loosen and tighten the collar is saved, as well as the time required for measuring the height of the arm by the miner. Standard 4-in. lagging (10.16 cm.) is used under the bar. It is found that the bar is jacked up an average of 2⅛ in. (5.4 cm.) if the screw is run into the bar before the bar is set up.

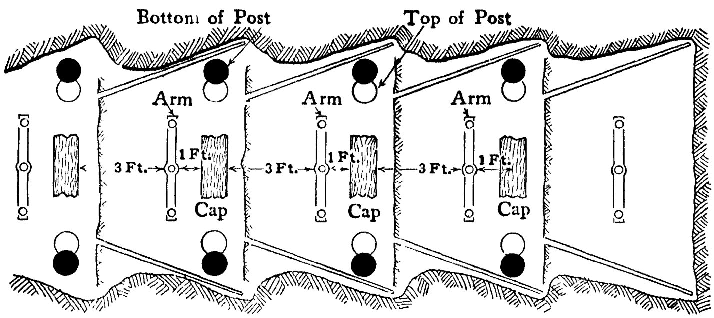


FIG. 19.—COÖRDINATION OF THE DRILLING TO THE TIMBERING IN A SHIFT.

Standard Rounds for Raises

Figs. 20 and 21 show the instruction card for the standard six-post raise; this is, of course, drilled with stoping drills. The extension of the machine is placed 12 in. from the post and 12 in. from the girt. The corner hole is drilled at an angle of 80° and at this point the bottom of an 8-ft. (2.44-m.) hole will be 6 in. (15.24 cm.) outside of the timber. The center side hole is drilled vertically and the cut holes at an angle of 78°. In this way all the holes are drilled from four positions, one at each corner of the raise. Three cut holes are drilled 9 ft. deep and should come within 4 in. of meeting at the bottom. Two center cut easers are drilled 4 ft. deep and should also come within 4 in. of meeting; these are drilled at an angle of 70°. In blasting, the primers are put in the center of the powder used or slightly higher.

Standard Rounds for Stopes

In stopes there are two faces to break to; consequently, all holes for a given stope are drilled with the same angle. In rill stopes, the holes

are drilled in rows parallel with the plane of the vein and at right angles to the strike. They are drilled 5 ft. (1.52 m.) or 7½ ft. (2.29 m.) deep according to the ground. Figs. 22 and 23 show the instruction card for

8 FOOT ROUND—SIX POST RAISE.

BE SURE TO EXAMINE BREAKERS BEFORE BLASTING

N.B. MAGAZINE MAN!!
SEE THAT THIS CARD IS COMPLETELY FILLED OUT.
DO NOT ISSUE EXPLOSIVES UNLESS BOTH THE SECTIONS ARE PRESENTED BY MINER
LOWER HALF OF CARD MUST BE HANDED BACK TO MINER FOR THE SHIFT BOSS!
RETURN THIS SECTION TO THE OFFICE!

5′10″ Cap	5′ Cap	Footage Blasted	
			CHECK!

Holes	1–2	3–4	5	6–7–8–9	10–11–12	13–14–15	16–17–18	19–20–21	22–23	24–25
Cut	To Center				None		From Center			
Steel	5′	10′	10′	10′	8′6″	8′6″	8′6″	8′6″		

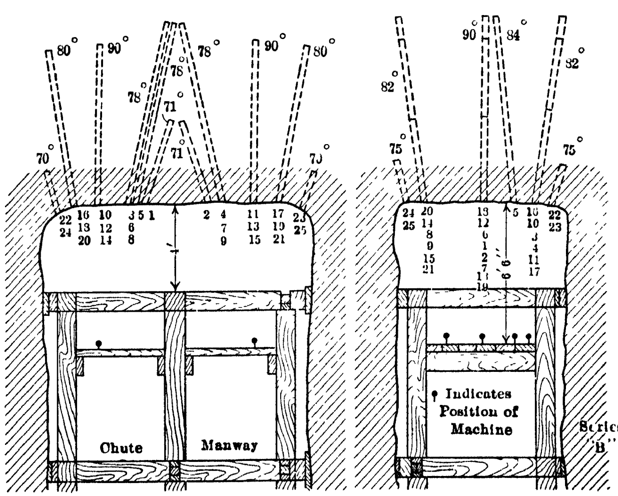


Fig. 20.—Instruction card for standard six-post raise.

square-set stopes, and Figs. 24 and 25 for rill stopes. These cards show the arrangement of the holes. They are drilled 4½ ft. or 8 ft. depending on the dip and width of the vein. Drilling is kept several sets in advance of the face of the stope. The advantages of this are as follows:

8 FOOT ROUND—SIX POST RAISE

FOR EX-HARD, HARD, MEDIUM, SHORT, AND SOFT GROUND.

DRAW A HEAVY LINE THROUGH KIND OF GROUND!

DATE............ ....EXPLOSIVES ISSUED
PLACE...... .. ..BOSS'S NO.... ....BEFORE LUNCH..............
MAGAZINE NO... ....MINER'S NO... ....AFTER LUNCH...............
TYPE OF DRILLING MACHINE...
USE TWO STICKS OF TAMPING TO A HOLE.
DO NOT ISSUE POWDER IF CARD IS DETACHED!

Waste Direct to Chute	Waste Direct to Gob	Feet in Ore	Feet in Waste

	Holes		Powder		Tamping	Primers
	Number	Depth	35%	60%		
Round						
Plugs						
Total						

Hole Numbers	Ex. Hard		Hard		Medium		Short		Soft	
	35%	60%	35%	60%	35%	60%	35%	60%	35%	60%
1–2	3	0	2	0	2	0	2	0	0	0
3–4–5–6–7–8–9	5	4	4	4	4	3	8	0	3	0
10–11–12–13–14–15	4	4	3	4	3	3	7	0	3	0
16–17–18–19–20–21	4	4	3	4	3	3	7	0	3	0
22–23–24–25	2	0	2	0	1	0	2	0	0	0
Total	97	76	76	76	72	57	152	0	57	0

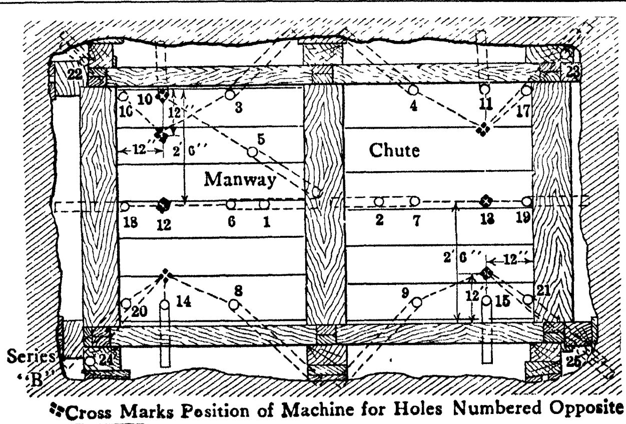
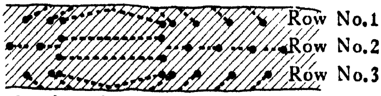


FIG. 21.—INSTRUCTION CARD FOR STANDARD SIX-POST RAISE.

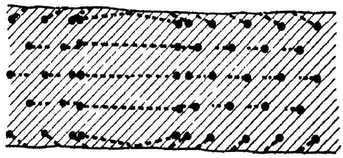
TIMBERED STOPES

Drilling and Blasting — 9 to 14 Ft. Vein

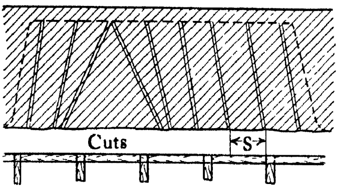
Case II (Over 70° Dip)



Spacing when Three Rows are used



Spacing when Five Rows are used



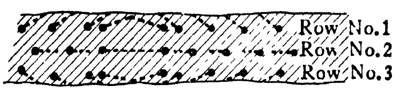
Section showing Slope of all Holes

Instructions

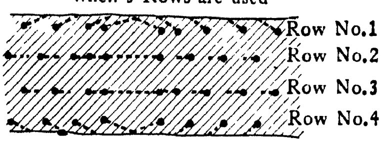
Holes - To be Drilled from Platform
" - Regular-Slope: 1-in 6 (Along Vein)
" - Cuts- " = 3 " 6 (" ")
" - " - To be Blasted with 60 % Gel.
Miner must put a 6 in. Plug on Footwall Side of the Last Hole he Drills for Identification Purposes.

Drilling and Blasting

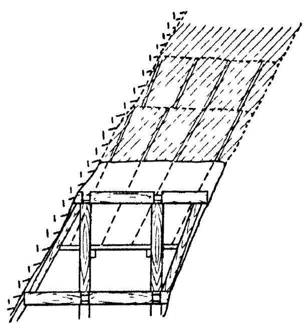
Case II (Under 70° Dip)



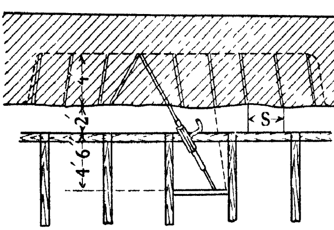
Plan - Showing Spacing of Holes when 3 Rows are used



Plan - Showing Spacing of Holes when 4 Rows are used



Case II
(Under 70° Dip)
End Section - All Holes in Ore



Section showing Slope of all Holes

Instructions

Slice No. I - To be Drilled from Platform
Slice No. II- " " " " Top of Set
" " - Spacing and Depth same as Slice I
Holes-Regular-To Slope 1 Ft. in 6 Ft (Along Vein)
" -Cuts- " " 3 Ft. " (" ")
" " - " be Blasted with 60 % Gelatin
Miner must put a 6 in. Plug on Footwall Side of the Last Hole he Drills for Identification Purposes.

"Keep the Ore Clean"

FIG. 22.—INSTRUCTION CARD FOR SQUARE-SET STOPES.

1. In case of a breakdown of compressor or drill or shortage of drillers for the stope, the stope is not short of ore, as holes already drilled can be blasted.

TIMBERED STOPES

DATE.............. 9 TO 14 FT. VEIN
PLACE...................... .BOSS'S NO....................... .BEFORE LUNCH ....
MAGAZINE NO. ... .MINER'S NO. .. . . AFTER LUNCH ....
POWDER ISSUED 60% ... .35% . .. .HOLES BLASTED.. ....
PRIMERS ISSUED 7 FT......... ...... 10 FT. . . .. PLUGS BLASTED ... .
TAMPING (use 4 sticks to a hole) FT. BLASTED { IN ORE . ...
DRILLING MACHINE USED .. .. IN WASTE .. . .
CHECK WIDTH OF VEIN | 9 | 10 | 11 | 12 | 13 | 14 |

N B. *Magazine Man.*—See that both sections of this card are completely filled out. Issue no explosives unless both sections are presented by miner Return this section to office.

N.B. *Shift Boss*—Check width of vein on both sections of card. Draw heavy line through character of vein. Do not detach sections.

WIDTH OF VEIN		Depth Hole Case I	Dist. from Waste	S Spacing	Case I EX-HARD	Case I HARD	Case I SHORT	Case I MEDIUM	Case I SOFT	Case II EX-HARD	Case II HARD	Case II SHORT	Case II MEDIUM	Case II SOFT	Depth Hole Case II
9-10.	Frozen walls	8′	6 in.	S	30	36	36	36	40	28	32	32	32	36	$4\frac{1}{2}'$
	Free walls		12 in.	S	36	44	44	44	46	32	36	36	36	40	
	Number of rows				4	4	4	3	3	4	4	4	3	3	
	Powder per hole			35%			5	5	5			3	3	3	
				60%	8	7				4	3				
11-12	Frozen walls	8′	6 in	S	32	36	36	36	40	30	32	32	32	36	$4\frac{1}{2}'$
	Free walls		12 in	S	36	40	40	40	46	32	36	36	36	40	
	Number of rows				5	4	4	3	3	5	4	4	3	3	
	Powder per hole			35%			5	5	5			3	3	3	
				60%	8	7				4	3				
13-14	Frozen walls	8′	6 in	S	32	36	36	36	40	30	32	32	32	36	$4\frac{1}{2}'$
	Free walls		12 in	S	36	40	40	40	46	32	36	36	36	40	
	Number of rows				5	4	4	3	3	5	4	4	3	3	
	Powder per hole			35%			5	5	5			3	3	3	
				60%	8	7				4	3				
					Case I (over 70° dip)					Case II (under 70° dip)					

PLUGS—Use one-half stick of 35% gelatin for each foot of hole
CUT HOLES—To be drilled 6 in deeper than regular holes
'ILLUSTRATIONS AND INSTRUCTIONS—See back of this card
'KEEP THE ORE CLEAN."

FIG. 23.—INSTRUCTION CARD FOR SQUARE-SET STOPES.

2. The driller works in a different part of the stope from the timbermen and shovelers; therefore, the men do not get in each other's way.

3. Drilling can be done in a more regular and systematic manner and the holes more accurately spaced.

4. The rock is sorted to better advantage and the ore is kept cleaner.

5. The drilling platform need not be taken down for blasting. The drill can be placed behind a timber set to be protected from flying rock.

RILL STOPES

DATE........ .... 2 TO 4 FT. VEIN
PLACE............. ....BOSS'S NO. .... .BEFORE LUNCH........... ..
MAGAZINE NO.... .. ..MINER'S NO. .. . . .AFTER LUNCH .. ..
POWDER ISSUED 60%.... . ...35% . .. . HOLES BLASTED .. ..
PRIMERS ISSUED 7 FT.. .. .. 10 FT. .... PLUGS BLASTED ....
TAMPING (use 4 sticks to a hole) . ...
DRILLING MACHINE USED....... .. . .
CHECK WIDTH OF VEIN | 2 | | 3 | | 4 |

WIDTH OF VEIN	*N.B. Magazine Man*—See that both sections of this card are completely filled out Issue no explosives unless both sections are presented by miner Return this section to office. *N B Shift Boss*—Check width of vein on both sections of card Draw heavy line through character of vein Do not detach sections.	Depth Hole Case I	Dist from Waste	S Spacing	Character of Vein: EX-HARD	HARD	SHORT	MEDIUM	SOFT	EX-HARD	HARD	SHORT	MEDIUM	SOFT	Depth Hole Case II
2	Frozen walls	7½′	3″	S	22	26	26	30	34	18	22	22	26	30	5′
	Free walls		6″	S	26	32	32	36	40	24	28	28	32	36	
	No of rows				2	2	2	2	2	2	2	2	2	2	
	Powder per hole			35%			4	4	4			3	3	3	
				60%	6	5				4	3				
3	Frozen walls	7½′	6″	S	22	28	28	34	40	20	24	24	30	36	5′
	Free walls		6″	S	28	34	34	38	48	24	30	30	36	42	
	No of rows				2	2	2	2	2	2	2	2	2	2	
	Powder per hole			35%			4	4	4			3	3	3	
				60%	6	5				4	3				
4	Frozen walls	7½′	6″	S	30	34	34	40	46	22	24	24	30	36	5′
	Free walls		9″	S	34	38	38	44	48	24	32	32	38	44	
	No of rows				2	2	2	2	2	2	2	2	2	2	
	Powder per hole			35%			4	4	4			3	3	3	
				60%	6	5				4	3				
					Firm Walls					Poor Walls					

PLUGS—Use one-half stick of 35% gelatin for each foot of hole.
ILLUSTRATIONS AND INSTRUCTIONS—See back of this card.
"KEEP THE ORE CLEAN"

FIG. 24.—INSTRUCTION CARD FOR RILL STOPES.

A standard drilling platform is employed and facilitates operations greatly. It enables the operator to work more conveniently and also saves on the length of drill steel. The height of the platform is so adjusted that when the machine is run out to full length the chuck is just

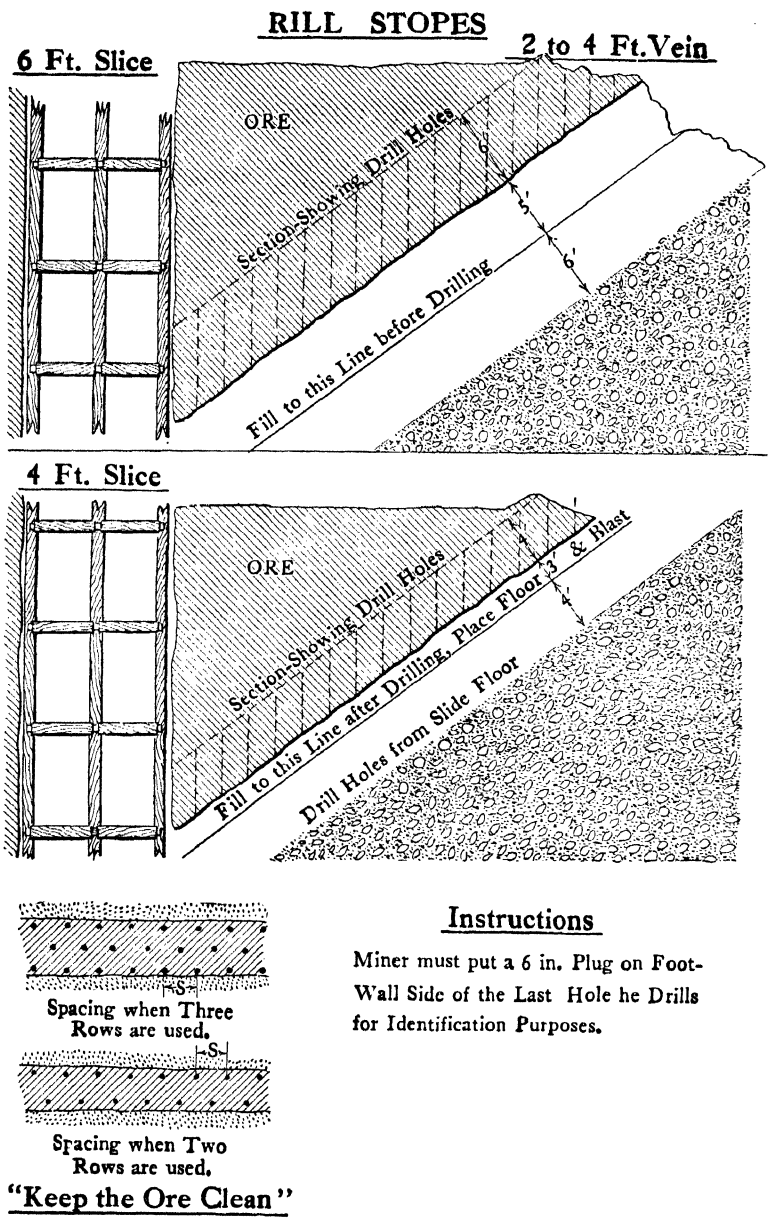


FIG. 25.—INSTRUCTION CARD FOR RILL STOPES.

up against the back. There is, therefore, no wasted length of drill steel. The platform is hung on hooks with sharpened points and U shape below, in which 5 by 10-in. lagging (12.7 by 25.4 cm.) is laid. The sharpened points of the hooks are driven into the posts, the lagging dropped in the U, and planks laid across. The platform is quickly built and can be moved, raised, or lowered, in a few minutes.

Ventilation

Ventilation practice at North Butte is planned to go considerably beyond providing the theoretical amount of air necessary to keep the supply pure. We are not dealing with explosive mixtures, as in coal mines, but with gases from blasting and rock dust from drilling, both of which are very injurious to the health of the miners; also with under-

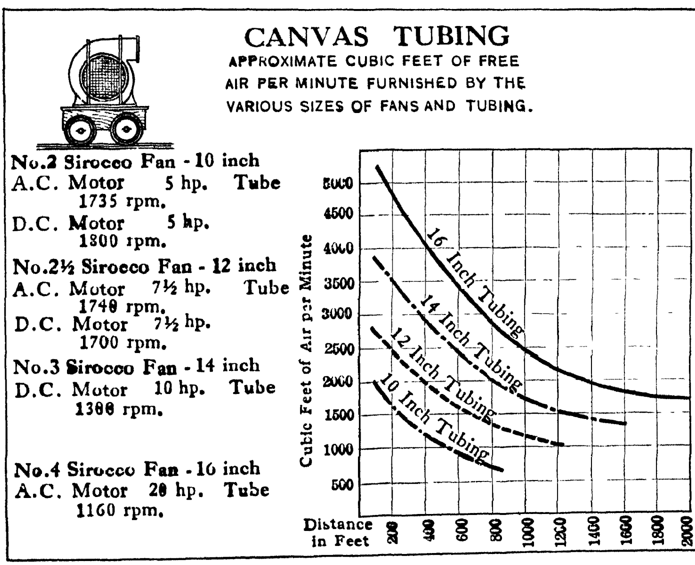


Fig. 26.—Chart showing approximate amounts of air furnished by various sizes of fans and tubing.

ground temperatures inherently high, causing the bodies of men who work in it to be enveloped in a heated, moisture-laden layer of air, which is also injurious. It is therefore necessary to blow a large surplus of air into most of the stopes and raises, and some drifts where the heat is excessive, in order to remove this saturated layer of air from around the bodies of the workmen and give them cool air to breathe and in which to work. In standardizing this work, it is simply a matter of using the portable fans and canvas tubing whenever and wherever conditions re-

quire. A complete ventilating outfit can be taken from the surface underground and be running in 3 hr., or it can be taken out and put on the surface in 1½ hr. The equipment is also very efficient for clearing gas out of muck piles, and since adopting it as a standard there have been no cases of miner's headache arising from mucking piles of rock that hold carbon monoxide in mechanical mixture.

Where fans are used, it is seldom necessary to run raises from stopes to the levels above. The fans give far better ventilation than natural ventilation through the raises. The raise manways and chutes have a high resistance to the flow of air and the amount of air that flows through them is small, whereas the air current from the fans is strong and positive. Where the stopes do not run through to the level above there is a large saving in expense.

Each installation of ventilating equipment is a separate problem, since there is so much variation in the widths of ore stoped and position of both stoping and development faces with reference to the main air supply. Calculations can be made of the volume of air to be displaced and the time required to displace it. This involves figuring the total cubic feet of the openings and an estimate of the excess flow required to reduce the heat or to meet other conditions. The fan equipment can then be selected of sufficient capacity to meet the requirements. Usually this is determined by practical experience rather than calculation. In Fig. 26 are shown the approximate amounts of cubic feet of air per minute that will be delivered with the different sized fans and tubing operating at varying distances.

Timbering

Timbering practice in the Butte mines has been standardized for a long time, and details were contributed to the Institute several years ago.[1] Since that time there has been practically no change. No improvements in timbering methods were indicated by the studies, but it was shown that about 40 per cent. of the timberman's time was spent looking for timber and additional time was lost looking for other supplies. This time has been reclaimed to a great extent by the better routing of all supplies. There are now timber depots on each level underground, in which an ample supply of timber is kept constantly on hand. The timber is all framed on the surface and stenciled with the number of the particular working in which it is to be used. Tally boards in the office show exactly what is on hand at each timber depot at all times, and as the stock is depleted below the standard it is automatically renewed without special orders being sent in by foremen or shift bosses.

[1] B. N. Dunshee: Timbering in the Butte Mines. *Trans.* (1913) **46**, 137. N. B. Braly: Shaft-sinking Methods of Butte. *Trans.* (1913) **46**, 151

In addition to the large timber depot on each level, a small timber depot is cut in at each raise. In this small depot are kept on hand twelve posts, six caps, eight girts, one crate pole, six cars assorted blocks, twelve bundles of wedges, forty chute lagging, thirty 4 by 10-in. lagging, thirty assorted chute braces, six 3 by 10-in. by 3 ft., six 4 by 10-in. by 4 ft. 1-in., ten 2 by 10-in. slide lagging, one keg 60-penny spikes, one keg 80-penny spikes, two platform hooks, one machine drill, and one steel wrench. As material is used from this raise, it is replenished from the main timber depot or from the surface.

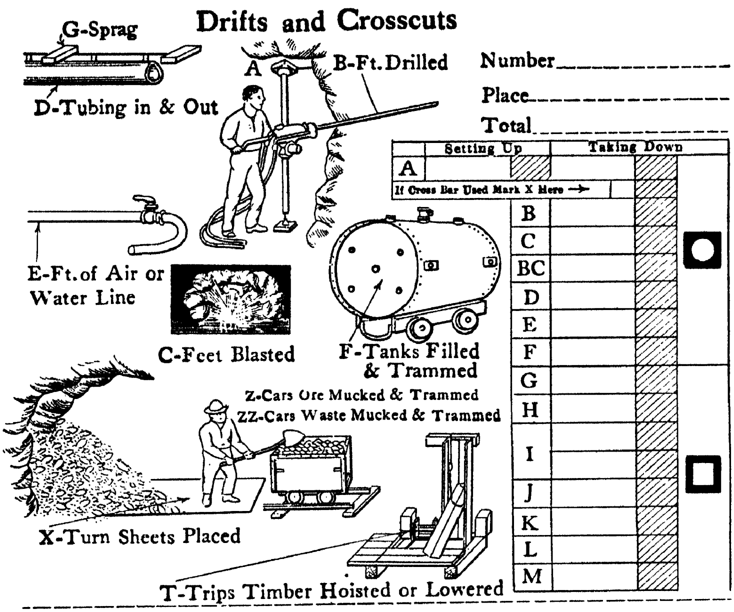
Standardization of Work

The application of time studies to mining operations on any comprehensive scale, the writer believes, is new. The value of time studies was first emphasized some years ago by Taylor and has been widely recognized in manufacturing and other productive industries, but aside from special cases has never been used in controlling or standardizing underground work. All time studies at North Butte were made by the company's staff. The well known timing methods in general use in manufacturing industries were employed, but it was considered best to have the actual work done by men of experience in metal mining and familiar with the particular underground conditions to be studied. The investigations were very comprehensive. Every operation underground was timed and a careful record was kept of the figures. The data, which were quite voluminous, after critical analysis by Mr. Braly and his assistants, formed the basis of the standard of efficiency ratings now in use. The time studies also suggested numerous improvements whereby individual work could be made more productive with little or no increased effort.

A classification of underground labor into various kinds of work performed under present conditions is as follows:

Miners, 15.7 per cent.; shovelers, 26.8 per cent.; trammers, 18.9 per cent.; timbering, 26.4 per cent.; powdermen, 2.3 per cent.; nippers, 2.1 per cent.; station tenders, 1.8 per cent.; shaft repairs and shaftmen, 2.4 per cent.; draining and ventilation, 1.9 per cent.; top carmen, 1.3 per cent.

The greatest difficulty in applying the results of these time studies to actual operations was to find a simple method of recording the work of each individual workman. This was finally solved by the "Pictorial" record card, shown in Fig. 27, devised by Mr. Braly. On this card, the letter A, for instance, represents setting up a machine and taking it down, including bringing up the machine from the drill platform to the breast, bringing the column to the breast and setting it up, placing the machine in proper position, connecting both air and water hose, oiling



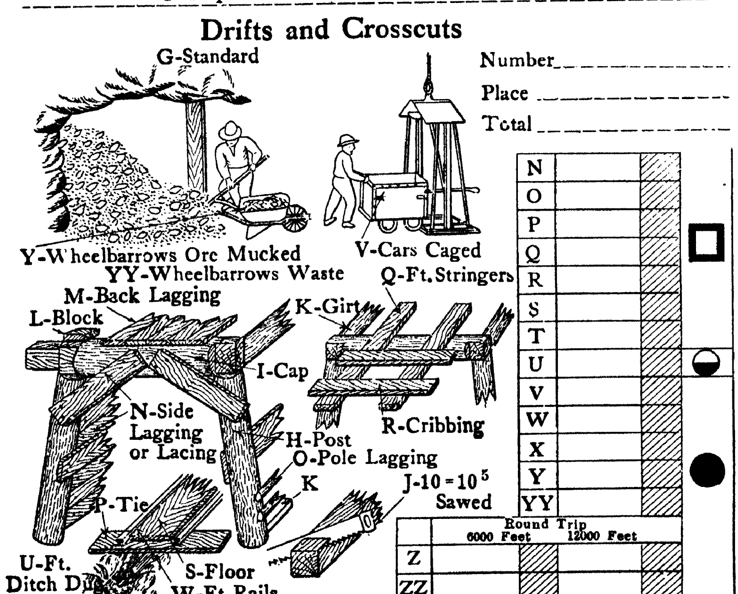


FIG. 27.—CARD FOR RECORDING WORK OF INDIVIDUAL WORKMEN.

the machine, etc., all preparatory to drilling. Should the time studies show that the proper time for two men to perform this is 40 min., if they perform it in 50 min. they are 80 per cent. efficient; if they perform it in 30 min., their efficiency would be 133 per cent. Every operation in drifting and crosscutting has been timed and is known, so that it is only necessary to record the operations done by any man underground for an entire day, add them together, and compare the totals with the standard in order to arrive at his efficiency for the day. Shoveling, tramming, timbering, and all other underground work were timed in the same careful and detailed manner.

In practice, the shift boss makes out cards daily for the men in his crew and sends them to the office, where the factors are applied. The cards for each man are totalled weekly and the factors have been so worked that the units can be added direct on a machine without involving any excessive amount of clerical detail.

These cards are not only efficiency records; they are instruction cards for the men and the shift bosses. They also give the shift bosses specific information as to what men in different positions ought to do; this is one of the most important results that has developed by the use of this system. Many shift bosses are inclined to give some men too much work and not enough to others, not intentionally, but because they have not crystallized out in their minds the exact amount of work a man should do. Further, there is recorded on the cards the supplies used each day in each working face. This facilitates compiling; and it has been found possible to improve considerably the routing of supplies to points of use, and thus eliminate time lost because the men did not have their supplies within convenient distance. Stocks and the location of underground supplies are kept track of on tally boards, on which supplies used each day are pegged up from the cards; the stocks are replenished without special requisitions from shift bosses. Tally boards are also used for compiling the output of ore and waste, rock shoveled, and other items of work done in the various working places.

Considerable time has also been saved by routing work through the shops. The system is adapted from commercial machine shop practice. When a new job is started, the master mechanic makes out shop orders for the various operations required, whether for the blacksmith, boiler, or machine shop and hangs these cards on racks, which have a row for each mechanic and his helper; the number tags are placed at the left hand end of the rows. The master mechanic arranges the cards so that the job is routed through the shops to the best advantage. No man runs out of work and the material is on hand as needed. Each man takes the cards in order as they are placed on the rack, but this order is frequently changed as the jobs progress through the shops. As each job is completed, the mechanic enters up his time and materials used and the card

becomes the shop record. This method of systematic shop planning has practically eliminated expensive rush jobs, except in case of breakdowns of machinery, and also cuts out delays to the mine in receiving needed equipment from the shops.

After having established the time standards and installed the system of recording individual efficiency, a bonus system was inaugurated under which underground men are paid a bonus based on efficiency rating in excess of 80 per cent. of the standard. All underground men are paid the standard rate of day's pay regardless of rating, but if they make a rating of over 80 per cent., they are paid at fixed rates per point above 80 per cent. This is an incentive for the men to study their jobs and coöperate in raising the efficiency of the whole organization. However, the primary object of standardizing the work is to educate the miner to conserve his effort and not waste it. Ordinarily it requires much less energy to do a thing right than to do it wrong, and the application of the standardized methods has demonstrated this. The standard methods introduced have resulted in increased efficiency, but it is believed that they have at the same time enabled the workmen to do their work more easily.

It was also found that inefficiency was frequently due to men not being placed to the best advantage. For nearly three years, employment has been in charge of the assistant superintendent instead of the individual shift bosses. He interviews all applicants for work and furnishes the various shift bosses, on their requisitions, with such numbers of men as they require. Men who are recommended for discharge or who wish to leave are obliged to report to him; in many cases such men are transferred and retained as employees. Of men so transferred, about two-thirds have made good in their new jobs. Since establishing this system, labor turnover has been reduced over one-half and the shift bosses have found that they get a much better force of men than when they hired their men by selecting from those who applied each day. The policy does not contemplate or tolerate discrimination against any man who is, by no fault of his own, unable to make a high rating; the policy is to give any man who is willing to work a chance to do so, to locate him where he can do his best, and to educate him in his work so that he can do it better.

Education

A most important factor in promoting efficiency is to carry on systematic and regular instruction in underground operations. For some time past there have been held classes of instruction to prepare men for employment as shift bosses. These classes are open to any underground man ambitious to advance. From time to time written examinations are held, the questions being, of course, questions pertaining to practical

underground operations. All shift-boss positions are filled from men who pass these examinations. Before starting work as a shift boss, each man spends two weeks underground with the mine foreman. He thus becomes familiar with the mine and has brought to his attention the best methods known for handling its particular problems. By educating the shift bosses in this way, a good foundation is laid for educating the men, for the shift bosses must largely be depended on for carrying out the training of the men under them. Special instruction in the use of tools and in mining methods is also given all underground workmen who desire it.

Health

Every effort has been put forth to make underground conditions as healthful as possible. The writer wishes it were possible to go a step farther and, by means of physical examination and medical advice, assist the individual workman to conserve and improve his physical well-being. Prejudice against physical examinations has made it impossible to accomplish anything along these lines as yet. It is a matter that cannot be made compulsory or effective without the cooperation of the men themselves, but it would be of tremendous benefit to the men if it were possible to carry out this plan.

DISCUSSION

Grant H. Tod, Livermore, Calif. (written discussion).—To carry out successfully such a program as was prepared by Messrs. Braly, Frink, and Linton, certain fundamentals must be present: (*a*) There must be in the minds of the management a sympathetic understanding of the reasoning power of the mind of labor; by this, the writer does not mean any form of parlor Bolshevism. (*b*) Labor turnover must be prevented; large numbers of steady employees are necessary to make standardization methods economically sound. (*c*) The property at which a standard form of operation is desired must have been developed to a point where continuous operation is assured.

Several factors contribute to the success of the work at North Butte. Men who make good are assured steady employment, which makes for stability and satisfaction. The management's interest in the employee is shown from the moment he gets on the ground. The room in which he waits to be interviewed by the employment manager is well lighted and ventilated and furnished with reading matter as well as pictures and models showing underground operations. The employment manager was evidently selected because of his understanding of human nature. When a man is hired, he is, if possible, assigned to the kind of work he is best fitted to do. When a man proves unsatisfactory to his boss, or wishes to

leave, he is sent to the employment manager and, where practicable, is assigned to some other work.

The number of men employed and the quantity of material kept in the mine have been reduced to a safe minimum. There is no unnecessary riding on the cage, and the time consumed in lowering and hoisting men and materials is well regulated. Everything has its place and appears to be in it most of the time. The mine tracks are in good repair and there is a noticeable absence of idle equipment blocking trackage, at any point.

The most conclusive evidence of the possibility and advantage of standardization work, in many mines, is the cost of producing copper at the North Butte under an abnormally high cost of material and labor.

The efficiency engineer and safety engineer use the camera underground to great advantage. Photographs of bad or dangerous practices are made in the course of inspection rounds and stereopticon slides of these photographs are shown at the regular weekly meetings of the mine staff and shift bosses. Photographs are also made and slides shown of good practices as well as clever or original work done by any man or boss in order that proper credit may be given to the individual responsible for the work. This stimulates good work to a marked degree.

Chas. A. Mitke,* Bisbee, Ariz. (written discussion).—The Copper Queen branch of the Phelps Dodge Corpn. during the past eight or ten years has spent considerable time and effort trying to work out the most efficient method of performing the various operations connected with mining. Classes for foremen and bosses were held, at which the various standards in process of formation were discussed, then the bosses developed instructors who were sent around to train the men. Later, afternoon and evening classes, which all the men were urged to attend, were instituted at which, in addition to the company standards, subjects related to mining were discussed by men who specialized in the different branches. The attendance and results of these classes have been most gratifying.

Various other companies in the Southwest are making investigations along similar lines, with the object of developing standards to meet their particular needs. The operations connected with drifting, raising, and sinking have been fairly well standardized by most of the large companies. The general method of introducing these standards is for an instructor to spend from four to six shifts in a working place instructing the miners. The stoping methods in the large porphyry copper mines are also pretty well standardized, individual standards being worked out to meet conditions. In the high-grade mines, however, such a variety of stoping methods are still in existence that a number of standards must be used,

* Mining Engineer, Phelps Dodge Corpn.

and in many instances they must be sufficiently flexible to permit of considerable modification.

In the field of mining equipment there is also room for improvement. The types and weights of drilling machines could be reduced and the sizes and classes of steel reduced to correspond; many mines might find it feasible to standardize on one kind of steel only. Other improvements would be the manufacture of all machines with a universal chuck and universal air connections and diameters of air hose. Mine cars can also be standardized. It is true that no one type and make of car is suitable for all mining operations, but the number of types in use can be greatly reduced.

Mechanical shovels will in time supersede the mucker underground in a majority of cases; much can be done toward evolving standard shovels that will be suitable for conditions in metal mines, where space is extremely limited.

Athens System of Mining

By S. R. Elliott,* E. M., Ishpeming, Mich.

(Lake Superior Meeting, August, 1920)

The principles of the caving system, as they apply to mining soft iron-ore deposits, are well known, as this method has been in use for many years. It is, however, necessary to give a general description of it before describing the Athens method.

The shaft and surface equipment should be placed at a sufficient distance from the orebody, so that the caving and pulling of the surface will not disturb the operations of the mine. Levels are started at the proper intervals, and when raises are completed, slicing commences at the top of the orebody and progresses downward. The necessary precautions are taken to avoid the occurrence of large open rooms and it is customary to break sufficient capping to form a mat to prevent heavy falls of ground from crushing through to the lower sublevels and endangering the lives of the workmen. Slicing is continued, and when sufficient width and length are undercut, the capping commences to cave, and in time a settlement occurs at the surface. As soon as this caving commences, the pressure on the timber increases; after it extends to the surface, the amount of pressure has a certain relation to the depth. In other words, the pressure increases with the depth, but not in direct proportion. At great depths, such as are found at the Athens, 2500 ft. (760 m.), the pressure might be so excessive that the cost of keeping working places open would be prohibitive.

The actual caving of the surface in a deep mine, with a great thickness of capping, is slow and often takes years to accomplish. After sufficient width and depth have been undercut, the capping starts to break away, but, through arching, large masses of rock fill the opening, support the capping, and retard the continuation of caving. Caving to the surface is, therefore, retarded until either a sufficient thickness of the orebody is removed, or the width and length of the undercut part have been greatly increased. I wish to emphasize this fact of slow caving, for it is on this principle that the success of the Athens system largely depends.

Assuming that work is begun at the top, in this case at an elevation of about 1500 ft. (456 m.); that there is a thick bed of sand saturated with

* General Superintendent, Cleveland-Cliffs Iron Co.

water, which, in certain areas, reaches a thickness of 150 ft.; that the body strikes about east and west and pitches on an angle of about 12° to the west; when sufficient width and length and depth have been removed, a settlement will occur at the surface and the water, and possibly quicksand, will find its way to the workings. As mining is continued on successive sublevels, which are constantly increasing in depth, additional capping will break away and the size of the hole at the surface will increase. As the depth increases the pressure on the timbers will increase until, at great depths, this pressure may become so great that it will be impossible to keep the main drifts open, to say nothing of working places in sublevels.

For the successful operation of the caving system, it is necessary to plan sublevels so that as soon as a gang completes its work on the top sublevel, there is a working place for it at a lower elevation. In other words, in a given territory the removal of the ore and the undercutting of the hanging wall is going on at several elevations, but the work on the top sublevel is always farther advanced than on the succeeding sublevels.

The Athens system is the reverse of the ordinary caving system. The ore, under the capping, at the greatest depth in the mine is removed first and the work gradually extends from the bottom to the top of the deposit. The shaft is sunk to its ultimate depth and drifting on the bottom, or 2500-ft. level, is started. As soon as this drift is far enough away from the shaft for another gang to work with safety, drifting is started on the 2400-ft. level.

When the orebody on the 2500-ft. level has been crosscut, as shown at *D*, in the accompanying illustration, raising is started, commencing at 1 and gradually working back toward 11. In the meantime, crosscut *E* has reached its limit and raising is started in 12, and is progressing back to 22. The work on the 2400-ft. level is only slightly behind and crosscuts *F* and *G* are driven. Raises from crosscut *D* are connected with crosscut *F* and are then continued to their ultimate height, which is the top of the orebody or the capping.

As soon as the first raise 1 reaches the rock, slicing is started. This gang works on a pillar, the northern limit of which is a vertical plane midway between raises 1 and 2 and the eastern limit is a plane running north and south midway between raises 1 and 12, and stepping down to the west on the same angle as the inclination of the raises. The same method of slicing is used as in the ordinary caving system. As soon as a few slices have been taken under the capping, sufficient rock is blasted to form a mat to protect the men, in the sublevel below, from heavy falls of ground. When the part undercut has increased sufficiently, the rock will fall of its own accord and blasting becomes unnecessary.

After a few slices have been taken around raise 1, work is started at the top of raise 2, and then at the top of raise 12. The method of mining

in steps, and gradually increasing the depth, is shown on the accompanying plan and sections. The plan shows one area mined from hanging wall to foot wall; the part of the crosscut D in this area has, therefore, been abandoned. As mining continues, all of the ore adjacent to the bottom level is removed and the level abandoned. The line of retreat for the ore on the foot wall is approximately parallel to the same line under the hanging wall; it is marked on the plan "Area mined under hanging."

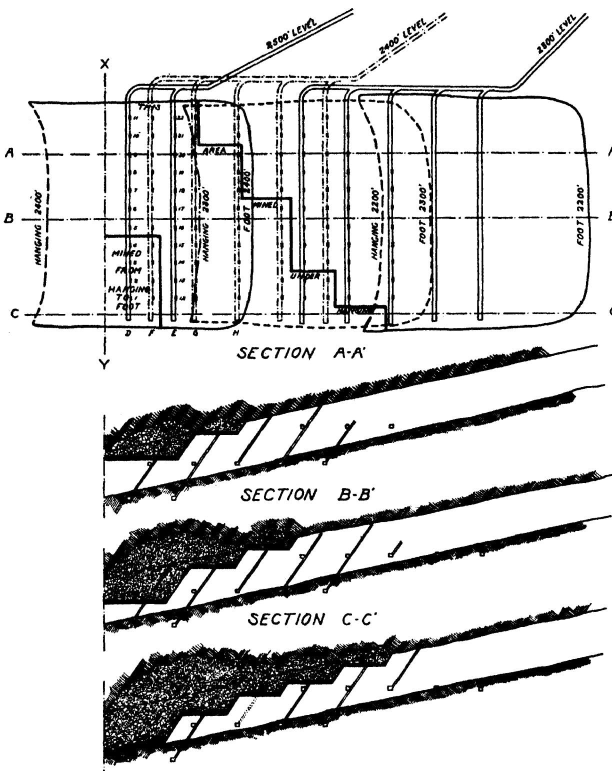


FIG. 1.—IDEAL PLAN AND SECTIONS.

Caving to the surface, as previously stated, is a very slow process, and at such a great depth as is worked at the Athens mine, we feel confident that a hole will not appear at the surface, until long after a large part of the ore has been completely removed from the hanging to the foot wall. Even if a cave does extend to the surface, it will be over an area that has been mined; it will be below the working places and not above them, as is the case in the ordinary caving system. In the ordinary system, the water

enters above the workings and is present during the life of the property, making the ore wet and giving bad working conditions. This is not the case in the Athens system.

The Athens system is practicable only in a comparatively flat deposit. The principal argument against its use is the interest on the investment. Against this argument we expect: (1) To avoid prohibitive pressures, thus making a large saving in timber cost. (2) The orebody will be drained and no surface water will pass through the working places, because if caves to the surface occur they will be below the workings. By draining the ore and providing dry working places the efficiency will be increased, and the cost lowered. The saving in freight, by shipping a dry ore, will amount to a large sum of money over a period of years. (3) Pumping will be less than in the caving system, because it will take years to break the surface, and we shall, therefore, for this length of time, only handle the water that is in the orebody. After the cave occurs and the size of the hole on the surface increases, draining a larger area, the head will constantly be decreasing, as successive levels are abandoned. (4) It is thought that the system is safer than the ordinary caving system, because a large proportion of the gangs are always working under a solid capping, and when a hole occurs at the surface it will be over a part of the mine that has been abandoned.

DISCUSSION

F. W. Sperr, Houghton, Mich.—As the angle of dip is about 12°, what difference would it make in retreating toward the shaft, in an orebody of this character, whether you started at the higher end and worked downwards or at the lower end and worked upwards?

Lucien Eaton,* Ishpeming, Mich.—As we do not own the land either east or west, the shafts must be north or south of the ore. The orebody strikes east and west and is in the Lucky Star property on one side and in the Bunker Hill property on the other. The shaft is on the north side. We went back as far as we could.

My idea of mining a very wide orebody of this character at great depth is to go to the side farthest from the shaft and mine out a vertical slice about 150 ft. (45 m.) wide for the full length of the orebody, treating this slice as though it were the entire vein. A slice 150 ft. wide and 2000 ft. long is large enough for almost any production. In this way, much of the development work and most of the timber roads will be in ore, and there will be little repairing in rock. When the first slice is finished, the weight of the broken capping will be carried on the foot wall, and the next slice can be mined without much difficulty. If the whole of the cap-

* Superintendent, Ishpeming District, Cleveland-Cliffs Iron Co.

ping over a wide orebody is undercut at once, the pressure becomes so great that there is difficulty in mining the ore at the bottom. In each vertical slice, the ore at the ends should be mined first, leaving the middle portion higher than the rest. Then the timber can be distributed downhill both ways and the air currents brought in in the opposite direction.

This method of mining is more a proposition of getting at the further boundaries of the property and retreating toward the shaft than of getting at the bottom of the orebody and working up. The only difficulty we have had in the application of the method is that the ore is 300 ft. high near the southwest corner and on the north it is about 100 ft. high, so that we could not start mining on the north side until we got to the top of the higher ore.

S. R. Elliott (author's reply to discussion).—If work is started at the high end of an orebody and retreats to the bottom, the well-known principles of the caving system are being used; if this method is reversed and you retreat from the lower end of the orebody to the top, the principles of the Athens system are used. The advantages claimed for the Athens system are enumerated in the last pargraph of the article. The time that has elapsed since the description was written has shown the value of the system.

Building Reinforced-concrete Shaft Houses

By J. Ellzey Hayden,* C. E., and Lucien Eaton,† M. S., Ishpeming, Mich

(Lake Superior Meeting, August, 1920)

The Cliffs Shaft mine of the Cleveland Cliffs Iron Co., located in the city of Ishpeming, Mich., is the largest producer of hard hematite ore on the Marquette iron range. The two shafts, *A* and *B*, lying 820 ft. (250 m.) apart on the east-west trend of the ore formation, were sunk in the early eighties, and, except for four years, have been steady producers of iron ore. Their location, on a hill overlooking the general offices of the company on the one side and the city of Ishpeming on the other, has made the shaft houses landmarks.

In the spring of 1919, as the old wooden shaft houses had become shaky and out of line, and would soon be unsafe, it was necessary to replace them with the least possible delay. Three plans of construction were, therefore, considered: To replace the old shaft houses by somewhat similar structures made of wood; to use steel and radically change the design; or to build reinforced-concrete shaft houses outside of the wooden structures, completely enclosing them. The first plan was disapproved, because it is anticipated that the mine will have a long life and the proximity of other buildings makes the danger from fire rather great. Delay in receiving material, and the necessity for closing the shafts during erection, were other factors in the situation. The second plan was disapproved on account of the high cost of steel, slow delivery, and interference with hoisting during the period of erection. The presence near one of the shafts of a large bed of hard gravel suitable for concrete made the cost of the third plan considerably less than the second, and little, if any, delay in hoisting was expected. Our expectations in this regard were fully realized. Furthermore, the work could be done by unskilled labor, which was plentiful, and the erection could be quickly started.

When the proposition was submitted to W. G. Mather, president of the company, he recommended that, on account of the prominence of the buildings due to their location, an effort should be made to combine in the design as far as practicable something of architectural beauty. Following his recommendation, the original plans were submitted to the

* Engineer in charge of construction, Cleveland Cliffs Iron Co.
† Superintendent, Cliffs Shaft Mine.

Condron Co., structural engineers, of Chicago, Ill., with George W. Maher, also of Chicago, as consulting architect. Three designs were submitted by them, of which that shown in Fig. 1 was chosen.

The two shaft houses are not exactly the same in design, being right and left hand and differing in several minor details; the plan and eleva-

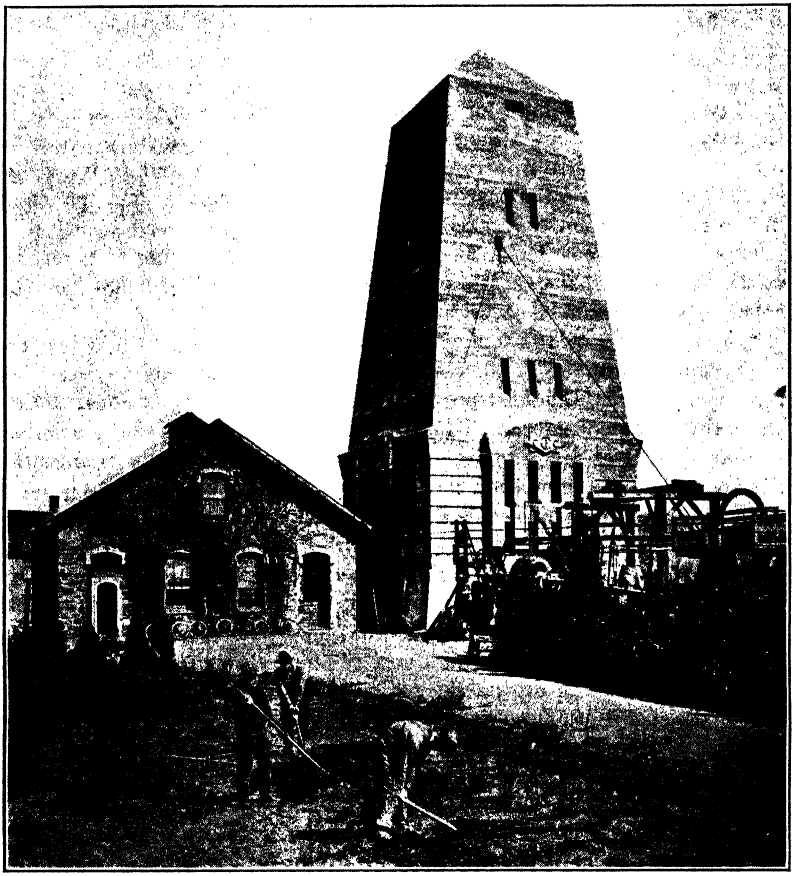


FIG. 1.—THE COMPLETED *A* SHAFT HOUSE.

tion of the *A* shaft house are shown in Fig. 2. The design calls for a reinforced-concrete building 33 ft. (10 m.) square inside at ground level, with solid vertical walls for 31 ft., then tapering to 21 ft. (6.4 m.) square at the eaves at a height of 88 ft. 9 in. (27 m.), with a pyramidal roof. The extreme height is 96 ft. 9 in. (29.5 m.) above the footings. Except where additional sheds have been built outside, there are fourteen win-

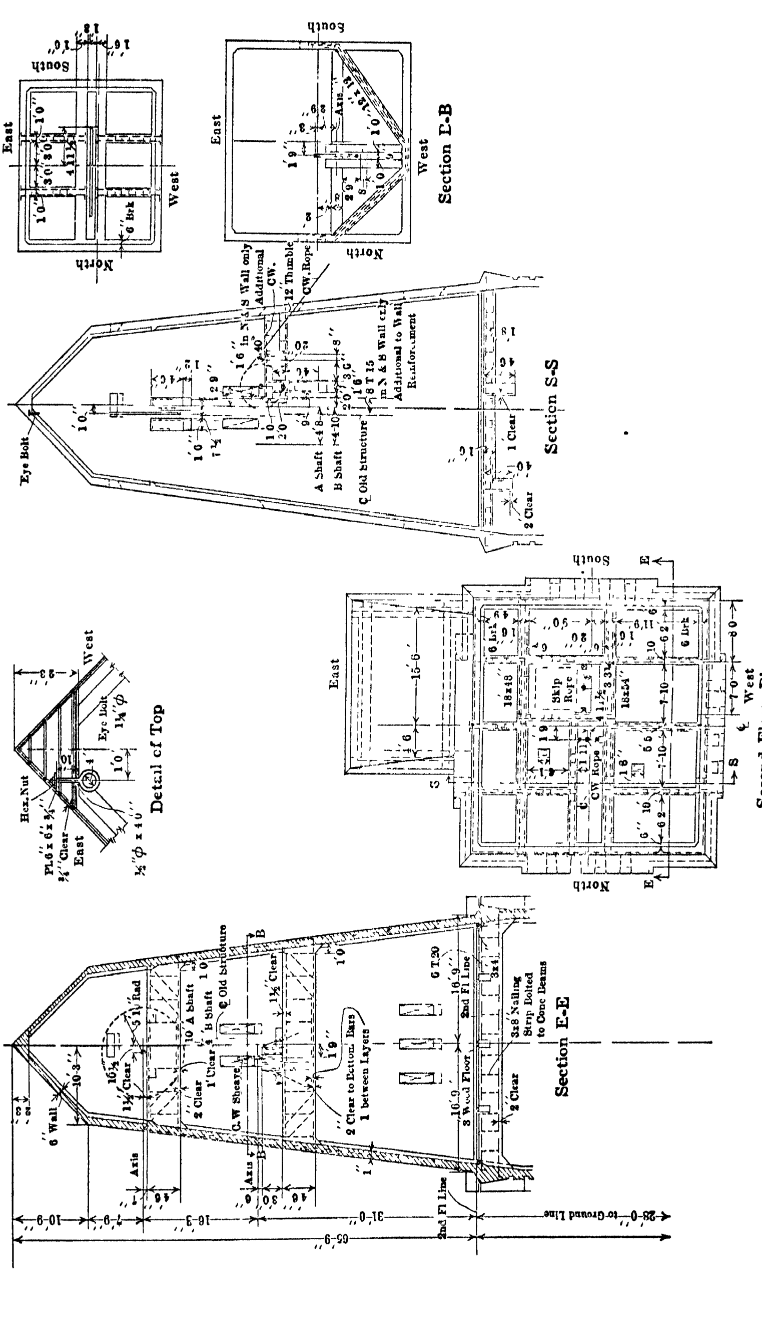


FIG. 2.—PLAN AND ELEVATION OF A SHAFT HOUSE.

dows on each side, and there are three doors. The three floors above the ground are connected by a ladderway. The pockets, built in 1910, in the old shaft houses were retained unchanged, and openings for them were left in the sides of the new shaft houses.

The shape and size of the new shaft houses had to be such that all new construction work would be clear of the old shaft houses, and the positions of beams and floors were governed largely by the available openings between timbers on the old shaft houses, as it was impossible to cut through any of these timbers. The beams for carrying the sheaves were heavily reinforced and made strong enough to carry a load that would break the hoisting rope. Extra reinforcing was put in the walls under these beams. The beams for the floors were designed for a safe load of 200 lb. (90.72 kg.) per square foot, and the floors were made of 3-in. (7.6 cm.) plank. The guides for the skip and cage are carried on vertical 12 by 12-in. (30.5 by 30.5-cm.) fir timbers, which are securely fastened to the concrete beams.

Work on the new shaft houses was started on July 21, 1919, both houses being built at the same time. The *A* shaft house was completed on Dec. 6, and required 725 cu. yd. (554.3 cu. m.) of concrete; the *B* shaft house was completed on Dec. 11 and required 1014 cu. yd. (775.3 cu. m.) of concrete. The work of tearing down the inside forms at both shaft houses was commenced soon after. A total of 132 working days was required for the work; 55 working days were required for pouring concrete and 77 working days for building forms and reinforcing. The maximum amount of concrete poured in any day was 52 cu. yd. (39.75 cu. m.).

The gravel for the work was obtained from a pit 200 ft. (61 m.) west of *B* shaft, where a bin was constructed, with a chute to feed the mixer for the *B* shaft house and a side chute for loading into wagons for transportation to the *A* shaft house. All gravel for the latter was hauled by teams. A 1¼-yd. (0.95 cu. m.) gravel car, hauled up an inclined track by an air-driven hoist, carried the gravel from the pit to the top of the bin, where it was passed over a 1¼-in. (3.2 cm.) grizzly, all oversize being raked off and wasted. From this bin, the gravel for the *B* shaft house was drawn off into a measuring box, which was tilted into the mixer by an Ingersoll-Rand Little Tugger hoist. The ½-yd. air-driven mixer was set below this measuring box and the concrete was drawn off into ½-yd. buckets, which were set on low trucks and trammed on a narrow-gage track to a skidway up the shaft house. At the *A* shaft, an inclined platform was constructed and the gravel wheeled in barrows to a 1-yd. mixer. A narrow-gage track carried the bucket to the skidway at the shaft house, as at *B* shaft. At both shafts, air-driven hoists installed on the second floor of the old shaft houses were used to raise the concrete to the hoppers, from which

it was run through wooden chutes to the forms. These hoppers were raised as the work progressed. In order to prevent separation of the aggregate, the vertical drop of the concrete into the forms was kept as low as possible, being seldom more than 6 ft. (1.52 m.).

At *B* shaft, the skidway was built outside the forms for the concrete shaft house, the upper end only being braced on the old building. When the top of the old shaft house was reached, a temporary framework of 6 by 8-in. (15.3 by 20.3 cm.) timbers was built 30 ft. higher and the skidway was connected to this. The legs that extended beyond the new roof were cut off when the concrete had been poured high enough in the forms. At *A* shaft, the skidway was built outside the old shaft house and inside the new one, and was carried up on a temporary framework to the roof of the new building. The concrete for the top part of the roof was dumped into a box and hoisted through a trapdoor in the roof by means of a 14-qt. (13 l.) bucket. This hoisting was done by hand, but less than 15 cu. yd. (11.5 cu. m.) of concrete was thus handled.

The ground around *A* shaft was hard and compact and offered no difficulty in placing the footings, which were put in to a depth of 3 ft. (0.91 m.). Two courses of 25-lb. (11.34-kg.) rail were placed in the footings, the bottom course of five rails, and the top course of three rails. A total of 119 cu. yd. (90.98 cu. m.) of concrete mixed 1 to 5 was used in these footings. At *B* shaft, considerable difficulty was encountered in placing the footings. The old shaft had been put down through 60 ft. (18.28 m.) of quicksand, which had formed a crater around the shaft during sinking, and this crater had been filled with rock, which was loose and full of voids. It was therefore necessary to go down to a maximum depth of 26 ft. (7.92 m.) to obtain a secure base for the footings. The ground under the old shaft house and around the shaft was then treated to a wet mixture of concrete, which closed the voids in the old rock fill. Three courses of 25-lb. scrap rail were placed in these footings, five rails in the bottom course, four in the middle, and three in the top course. Vertical rails were also set into the footings to tie them to the shaft house proper. A total of 339 cu. yd. (259.2 cu. m.) of concrete was put in for the footings of this shaft house.

The lumber used for the forms was as follows: 2 by 6-in. (5.08 by 15.24 cm.) hemlock studs were used throughout and were placed 16 in. (40.64 cm.) between centers. For the first 10 ft. (3 m.) above the footings, 2-in. rough hemlock plank was used, but above this 1-in. (2.54 cm.) hemlock, dressed on one side, was used; the change from rough to dressed lumber was made for the architectural effect. The lumber was given a coat of paraffine oil before being placed in the forms.

No. 9 annealed iron wire was used throughout in holding the forms. It was placed on every studdle at intervals of 2 ft. (0.6 m.) in elevation. Patented "EZ" clamps were used to draw up the wire and hold the forms

to correct dimensions. Deformed steel ½ in. (1.27 cm.) square was used throughout for the reinforcing, except over the doors and in the beams, where 1⅛-in. (2.86 cm.) square twisted steel was employed. The reinforcing was tied together and held in place with 6-in. (15.24 cm.) wire ties. Patented auger twisters were used to tighten these tie-wires.

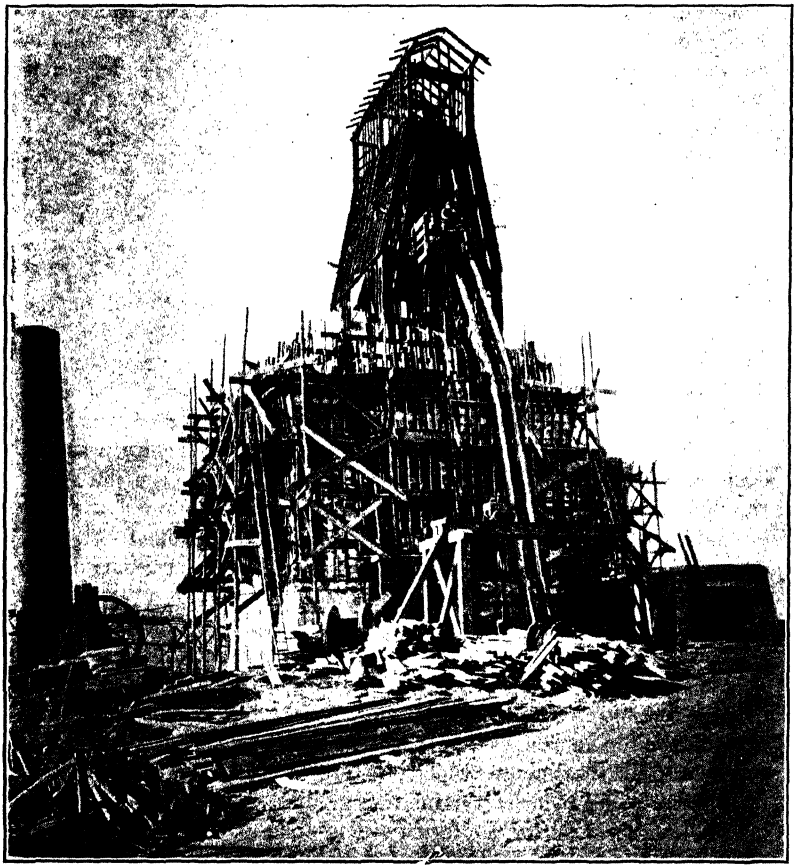


FIG. 3.—*B* SHAFT HOUSE UNDER CONSTRUCTION.

Huron Portland cement was used throughout the work, a 1 to 5 mixture being used in the footings, and a 1 to 4 mixture in the shaft house proper.

The shaft houses were built and poured in sections, varying from 6 ft. (1.83 m.) to 16 ft. (4.88 m.) in height. While the forms at one shaft were being filled, those at the other were being extended. On the entire job, hoisting in the shafts was interfered with only once; that was

for 4 hr. one Saturday night when it was necessary to stop the skip while the beams in the *A* shaft were being poured.

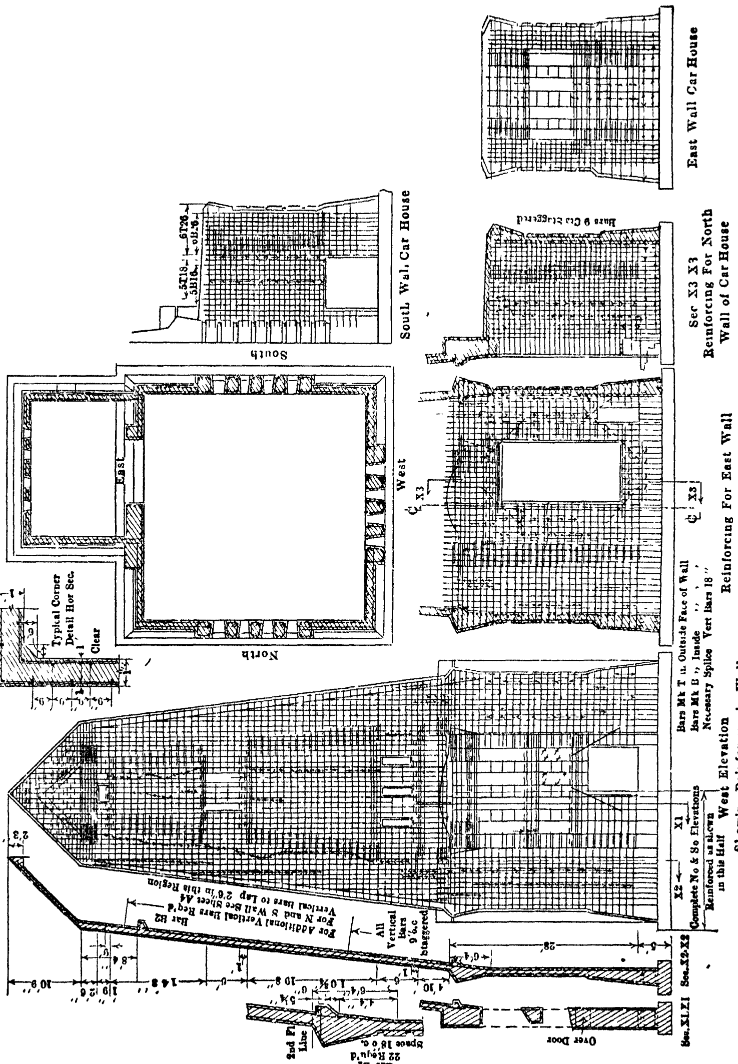


FIG. 4.—REINFORCEMENT FOR *A* SHAFT HOUSE.

The many architectural features of the design, together with the fact that it was impossible to brace the new construction on to the old shaft houses, on account of their constant vibration during hoisting, made the

work of form building difficult and slow. To help hold the forms to correct dimension during construction, concrete block spreaders were used at 6-ft. intervals. These spreaders, together with the "EZ" clamps, permitted the tightening of the form wires so that all heaving of the forms during pouring was eliminated. Great care had to be exercised in the distribution of the concrete into the forms, to prevent the weight from coming unevenly on the forms and pushing them out of line.

At first the pouring of concrete was done on both the day and the night shifts, in order to reduce to a minimum the number of lines in the walls. When cold weather set in, the height of the pour was reduced to such an amount as could be successfully completed on the day shift. After each pour the top surface of the concrete was mauled and roughed up, all refuse was blown off by a compressed-air blast, and before the new pour was commenced the top surface was washed with a mixture of cement and water. Also, when cold weather set in, each mixer was provided with a Hauck kerosene blower, which delivered a hot flame into the aggregate as it was being mixed, so that the concrete was delivered into the forms at a temperature of about 80° F. (27° C.). Large canvas tarpaulins were hung 8 in. (20 cm.) away from the form line, on the outside of the shaft, to reduce radiation over the section to be poured, and burlap bags were used for covering the top surface of the completed pour.

The very cold weather that existed throughout the last three weeks of the work necessitated further precautions against freezing. Steam radiators were installed on the inside near the top of each shaft house; the outside forms were packed with straw between studdles; and a second line of boards was put on. In addition to this, tarpaulins were hung on the outside. When the thermometer was at zero, the concrete 24 hr. after pouring was warm to the touch.

The crew employed consisted of fifteen carpenters, including one boss carpenter; nine reinforcing men, divided into three gangs of three men each; and two puffer men; in addition, twenty-four men were used only when pouring was being done. A blacksmith and helper were used part of the time for cutting and bending bars. The job was carried to completion without a serious accident, only a few minor cuts and bruises being sustained. The monograms placed on the panels near the cornice were cast separately in a mold, and were put in place after the rest of the building was completed.

The accompanying table shows the distribution of costs.

Cost of Erecting Two Concrete Shaft Houses at the Cliffs Shaft Mine, Ishpeming, Mich.

	Total			Per Yard		
	Labor	Supplies	Total	Labor	Supplies	Total
Concrete:						
Stripping gravel pit	$273 84	$334 68	$608 52	$0 158	$0.192	$0 350
Loading and hauling gravel	2,584.30	582 83	3,167 13	1 486	0 335	1.821
Cement	516 80	5,065 31	5,582 11	0 297	2.913	3.210
Forms, erecting and tearing down	13,411 99	7,185 71	20,597 70	7 712	4 132	11 844
Excavation and grading	511 93	112 55	624 48	0 295	0 064	0.359
Mixing and placing	4,821 05	743 92	5,564 97	2 772	0 428	3 200
Dressing surface	49.45		49 45	0 028		0 028
Special tools	12 79	20 30	33 09	0.007	0 012	0 019
Preliminary construction	2,225 14	404 26	2,629 40	1 279	0 233	1 512
Reinforcing	4,924 83	3,401 51	8,326 34	2 833	1.956	4 789
Total	$29,332 12	$17,851 07	$47,183 19	$16 867	$10 265	$27 132
Sheaves, floors, and runners	$3,250 90	$1,040 77	$4,291 67	$1 869	$0.599	$2 468
Lighting and heating	246 11	169.61	415 72	0 141	0 098	0 239
Tearing down old shaft houses	762 59	19.51	782 10	0 439	0 011	0.450
Moving tracks and chutes	620.34	8 01	628 35	0.356	0 005	0 361
Office, supervision, and consulting engineer		3,242 45	3,242 45		1 865	1 865
Total	$34,212 06	$22,331 42	$56,543 48	$19 672	$12.843	$32 515

DISCUSSION

William Kelly, Vulcan, Mich.—A distinction should be made between a shaft house and a headframe; this is distinctly a shaft house.

On the last page it is stated that "When cold weather set in, each mixer was provided with a kerosene blower, which delivered the hot flame into the aggregate as it was being mixed, so that the concrete was delivered into the forms at about 80°." How could the temperature of these materials be raised considerably by the flame in the short time the materials are in the mixer, without having the flame so hot that it would cause trouble?

Lucien Eaton.—The burner, which is placed over the discharge end, sends into the mixer a flame about 4 in. in diameter and from 20 to 30 in. long; it burns about 12 to 15 gal. of kerosene a day. The flame impinges directly on the mixture as it goes around. If the mixture gets too hot, there is a tendency for the aggregate to separate. The burner did not interfere with the discharge, nor did it injure the mixer, as the latter was turning all the time. We also heated the water with steam.

T. L. Condron, Chicago, Ill.—In the reinforcing of a structure, two elements are important; one is the reinforcing for the stresses due

to the heavy loads, which is the usual problem of making the beams, girders and supports ample for the loads they are to carry; the other, which is more important, is reinforcing for the temperature stresses. Concrete has a tendency to crack. Structures that are built of cut stone also crack, but the cracks develop in the mortar joints which is not so bad. In a concrete wall that is not reinforced, the cracks occur wherever the concrete may be weak, so the problem is to so reinforce the wall as to care for shrinkage stresses and the temperature stresses that will be repeated season after season. Mr. Eaton says that so far no temperature stresses have developed in these structures.

Lucien Eaton.—At the *B* shaft, which is the one to the west, there are two cracks in the shed that was built over the runway of the car from the crusher building. This shed was built after the shift house was started, and the foundations are not tied together; moreover, at the lower end, the water runs down the incline and gets into the ground and freezes. As those foundations are not tied together at the bottom, the frost raises the foundations of the shed in the winter time.

We knew pretty well how to reinforce to take care of loads, but reinforcing to prevent shrinkage was an entirely different proposition, and that was where we learned most from Mr. Condron. The idea is to put in the reinforcing in such small units and so close together that there is no room for cracks to develop.

Hoisting Equipment at Utah-Apex Mine

By J. A. Norden* and A. R. Willson,† Bingham Canyon, Utah

(Lake Superior Meeting, August, 1920)

Much has been written concerning the hoisting equipment of various mines throughout the country, but most of the literature on the subject, if not all, describes equipment of extraordinary capacity and the small and medium-sized outfits have been neglected. The hoisting equipment here described is capable of handling about 1200 tons per day. It has not been worked to capacity, but 600 to 700 tons, together with men and materials, have been handled in two shifts without any great effort.

Only one shaft extends from the surface to the lowest levels of the mine; see Fig. 1. It is composed of three compartments, two of which are used for hoisting, and the third for water, air, and electrical lines, and for sinking purposes. In 1909, this shaft was raised from the 1000-ft. (304-m.) level to the 700-ft. There a station was cut and an air hoist installed. At this time all mining operations were confined to the upper levels of the mine; that is, above the 700-ft. The ore was dropped to this level through chutes from the 300-, 400-, 500-, and 600-ft. levels and then lowered in cars on cages to the 1000-ft., which is the main haulage adit, the mill and railroad bins being about 3000 ft. from the shaft.

Orebodies were, in due time, developed on the 1000-ft. level and below. Therefore, in 1913, the shaft was sunk to the 1150-ft. level and thereafter sinking was continued at regular intervals until the 1500-ft. level was reached. But, hoisting from the 1300-ft. level was a difficult and slow process with the antiquated air hoist, so, in 1916, it was decided to raise the shaft to the surface and install a modern, electrically operated hoist. This was completed early in 1917 and the hoist was in operation in March of that year.

The Hoist

Figs. 2, 3, and 4 show various features of the hoist and electrical equipment. A Nordberg engine of the double-drum, geared type, designed for a 10-ton load, including rope, at a speed of 1500 ft. per min., is used. The drums are 7 ft. (2.1 m.) in diameter, with a 54-in. (137-cm.)

* Assistant Superintendent, Utah-Apex Mining Co.

† Master Mechanic, Utah-Apex Mining Co.

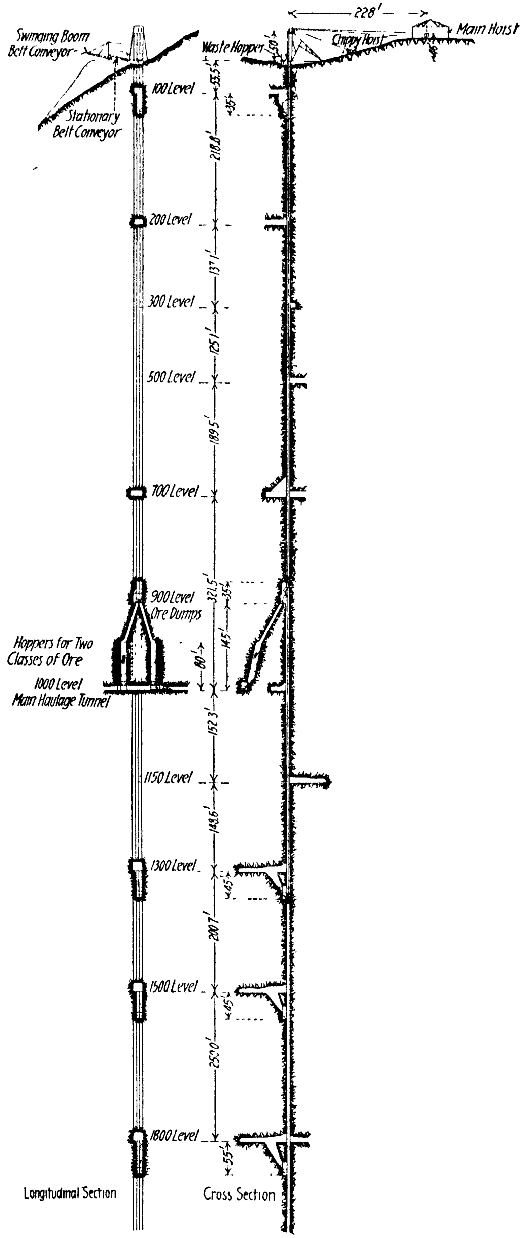


FIG. 1.—SECTIONS THROUGH PARVENU SHAFT SHOWING STATIONS, ORE POCKETS AND DUMPS.

face, and have a capacity of 3000 ft. (914 m.) of 1⅛-in. (2.9-cm.) rope. The gears are cast steel, with cut teeth of the herringbone type. The brakes are of the gravity-post type and the clutches are the axial, friction-plate design.

Both brakes and clutches are operated by hydraulic thrust cylinders, the brakes being applied by means of deadweights and released by the cylinders, while the clutches are applied and released by double-acting cylinders. The motive power is oil under pressure, furnished by a motor-driven accumulator. Hydraulic valves, governed from the operating platform by a system of levers, admit and exhaust the oil from these

FIG. 2.—VIEW OF HOIST FROM LEFT SIDE. TAKEN SHORTLY AFTER INSTALLATION.

thrust cylinders so that their movement absolutely follows the motion of the operator's levers, giving exactly the same control over the brakes and clutches that could be had by direct hand operation.

The accumulator consists of a cylinder containing a weighted plunger to give the desired oil pressure; its capacity is five brake applications. Oil is supplied by two motor-driven triplex pumps.

The hoist is operated by a 500-hp., 16-pole, wound-rotor, 450-r.p.m. motor, which drives through a single reduction of 19 to 129. Speed is regulated by the variable resistance of a rheostat of the liquid, weir control type. A sodium-carbonate solution is used as a conductor; its regulation is described later.

The current supply is three-phase, 60-cycle, 44,000 volts, which is stepped down to 2300 volts by a bank of three 250 kv-a, outdoor-type transformers. These are protected by lightning arresters, choke coils, fuses, and rotary air-break switches. From the transformers, the 2300-

volt lines are carried to a bus and switchboard in the hoist room. It has been found that the average load for starting a loaded skip is about 300 amp. for a few seconds, afterwards the full-speed running current is 75 to 125 amperes.

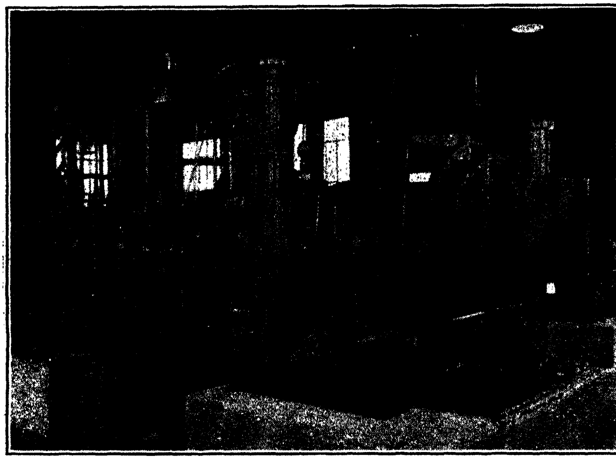


Fig. 3.—View of hoist from right side. Taken shortly after installation.

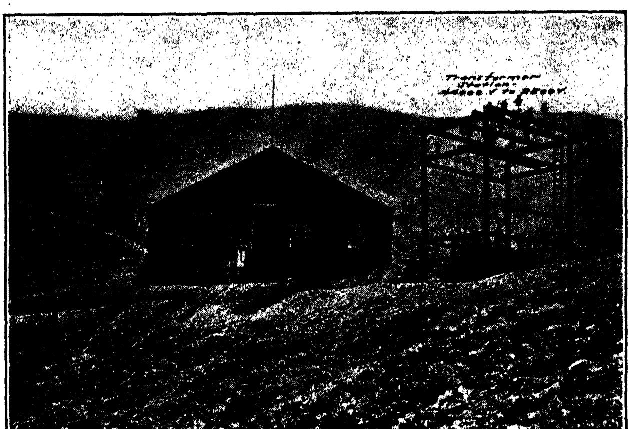


Fig. 4.—General outside view from rear. Taken before installation of lightning arresters.

Hoist Control

The entire hoist is controlled by three levers on the operating platform. Two of these operate the brakes and clutches, the third controls the motive power. This lever is mechanically connected to the controller and

the rheostat weir. A movement of a few inches forward of this lever engages the "forward" controller finger 4, Fig. 5, which closes the forward set of solenoid-operated contactors. This energizes the motor stator for forward rotation. As the lever is pushed farther forward, the rheostat weir raises the solution around the electrodes, which are connected across the rotor slip rings. The resistance is thus gradually lessened and the motor is brought up to speed.

When the electrolyte in the rheostat reaches a certain height, it covers a liquid contact. This closes a contactor whose fingers are in series with the operating coil of the rotor short-circuiting contactor, which allows this contactor to operate when the operating lever is thrown to the extreme position engaging finger 5, thus short-circuiting all rheostat resistance and bringing the motor to full synchronous speed, less slip. The function of the liquid contact is to prevent the premature short-circuiting of the rotor, in case the operating lever should be thrown to full-speed position before the rheostat allows motor acceleration. A reverse rotation is obtained by a backward movement of the same operating lever.

Hoist Safety Devices

The oil switch between the bus and the hoist circuit is equipped with overload relay and low-voltage release trips, to open the hoist circuit. These operate in case of a dangerous overload or a too great drop in voltage and open the emergency solenoid circuit, which sets the brakes and throws the controller to neutral position. In addition, several other auxiliary devices open this circuit, see Fig. 5.

A quick emergency stop may be effected by the operator by opening the master switch; this opens the main oil switch and causes the action above described. The low-voltage contactor and interlock contactor are auxiliary contactors, so connected with the control circuit that it is impossible for the motor to start when the power comes on after an interruption, if the control lever has not been left in neutral position. The two main contactors for forward or reverse rotation are mechanically interlocked so that it is impossible to operate one while the other is in the operating position or to operate both simultaneously. For overwinding, underwinding, and overspeeding, the Lilly hoist controller operates the emergency solenoid. In addition, limit switches are set on the head-frame and in the 900-ft. skip dump. These prevent overwinding by opening the emergency solenoid circuit; Fig. 5.

Skips and Pockets

With this new equipment, hoisting was continued with single-deck cages and cars. Through the years 1917 and 1918, years of high prices and intensive mining, the tonnage was crowded to the capacity of the sys-

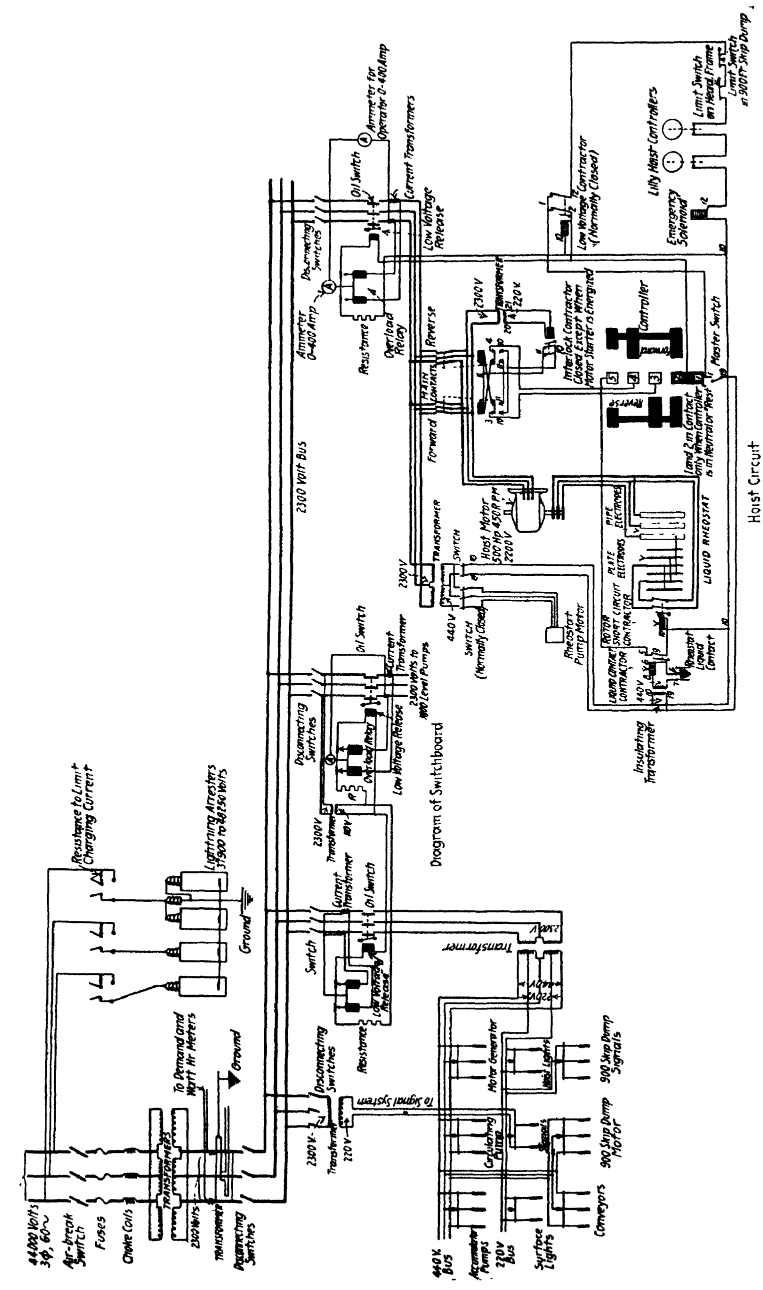


FIG. 5.—DIAGRAM OF WIRING FOR ELECTRICAL HOIST CONTROL AND SWITCH BOARD.

tem, which seemed to be about 700 tons per day of ore and waste; if anything went wrong the entire mine was soon held up for there was no storage capacity. During this period the installation of skips and ore pockets was first contemplated, but work was not commenced until

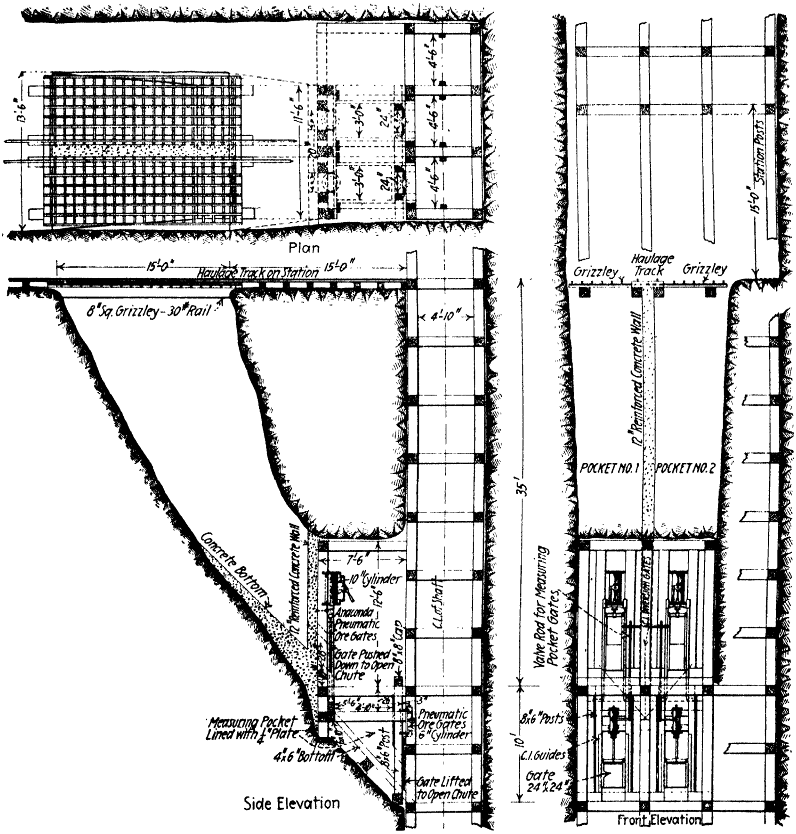


FIG. 6.—DETAIL OF ORE POCKETS FOR PARVENU SHAFT, SHOWING THE INSTALLATION OF PNEUMATIC GATES.

February, 1919, when, taking advantage of the low price of lead, ore production was curtailed. The work was completed in May.

At the present time mining operations are in progress on five levels, the 100-, 1150-, 1300-, 1500-, and 1800-ft. For all of these, except the 1150-ft., ore pockets were cut. Fig. 1 illustrates the position of these other stations along the shaft.

Three classes of material are produced on most of the levels: First-class or direct-smelting ore, second-class or milling ore, and waste. Hence it was necessary to provide for handling at least two kinds of

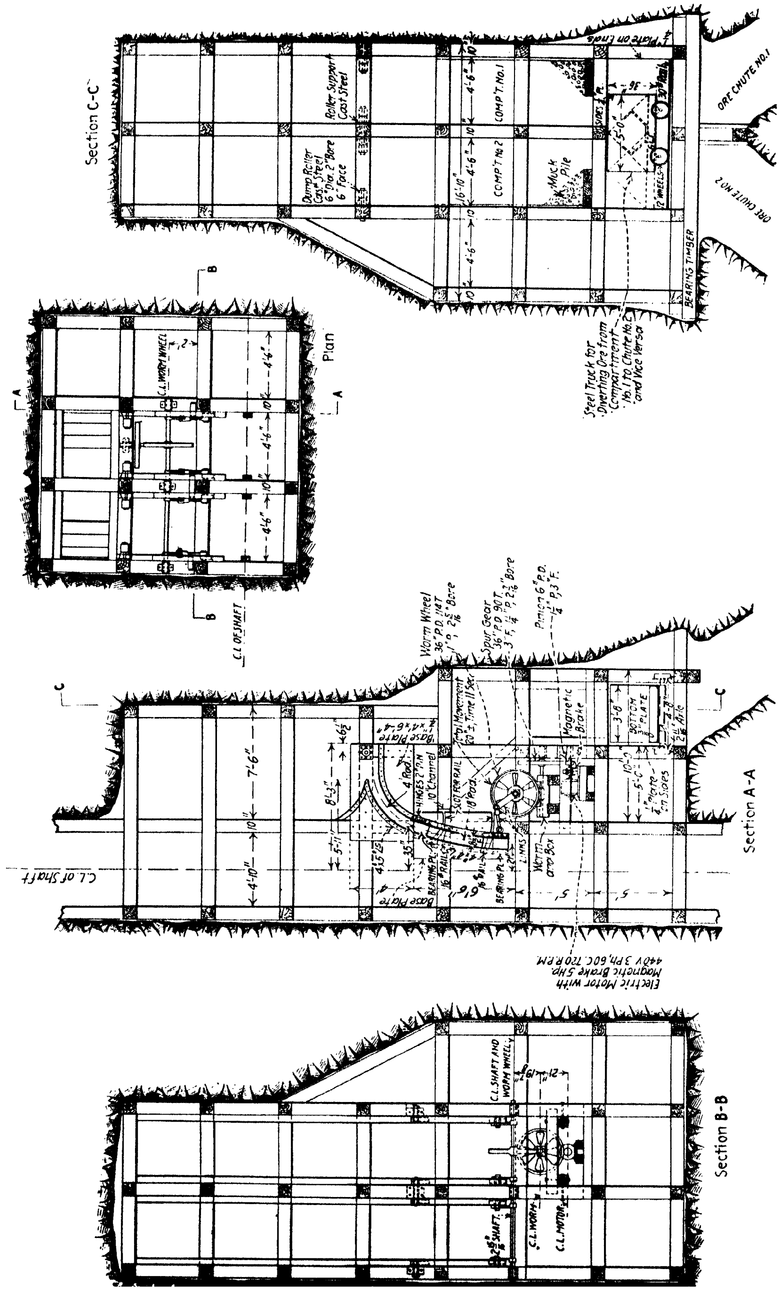


FIG. 7.—DETAIL OF SKIP DUMPS, DUMP GUIDE-MOVING MECHANISM AND DIVERTING TRUCK AT 900-FT. LEVEL OF PARVENU SHAFT.

material simultaneously at the shaft. The 100-ft. level, being an adit tunnel, has its own waste dump but more or less waste is hoisted to the surface from all of the other levels.

Shaft Storage Pockets

Fig. 6 shows the details of a typical ore pocket cut on the lower levels. It is divided into two parts by a reinforced-concrete wall. The ground is extremely hard, requiring no timber, and a small pillar was left between the main storage pocket and the shaft. The main gates are of the pneumatic type, designed and built by the Anaconda Copper Mining Co. The cylinders are 10 in. (25 cm.) in diameter and are equipped with a special control valve and check valves at each end to cushion the blow of the piston. Below the storage pockets are measuring pockets, which hold one skip load. These also are furnished with pneumatic gates, which have 6-in. cylinders, with an ordinary fourway stop cock for control, and angle-ball check valve at both ends to cushion the blow of the piston. These gates were designed and built at the mine.

Later, it was found to be desirable to divert the flow of rock from one pocket to the skip operating in the other compartment; this is accomplished by a swinging diversion gate between the measuring pockets, balanced with weights so that it is easy to handle. The top of the storage pocket is covered by an 8-in. (20-cm.) grizzly made of 30-lb. (14-kg. per m.) rail.

The 1300-ft. and 1500-ft. level pockets are identical, but the 1800-ft. pocket is slightly larger, being 10 ft. longer and proportionally larger at the top. On the 100-ft. level, however, due to soft ground, it was necessary to build the entire pocket in direct connection with the shaft. The measuring pockets, gates, etc. are standard with those of the other pockets.

The 1000-ft. Level Storage Pockets

All ore is dumped at what are known as the 900-ft. level dumps. From here it goes through chutes to the large receiving pockets of the 1000-ft. level; see Fig. 1. When these pockets were planned, it was the intention to use no timber, but it was soon found that the ground would not stand without support, so reinforced concrete was used.

The original design called for a 10 by 10-ft. (3 by 3-m.) pocket without a manway. Four regular stope sets would require an 11 by 11-ft. cut; this would do nicely for the pocket and as a manway would be very desirable, it was decided to carry up five regular stope sets, the fifth being for the manway. The cut was made large enough to allow 14 in. of concrete all around the pocket and on the manway side a 12-in. wall was made. For reinforcements, old rails were placed so that they formed 30-in. squares all around the pockets. Peep holes were left at regular intervals along the manway. After the concrete had set, the four stope sets of the pocket were withdrawn, and the fifth or manway set was left

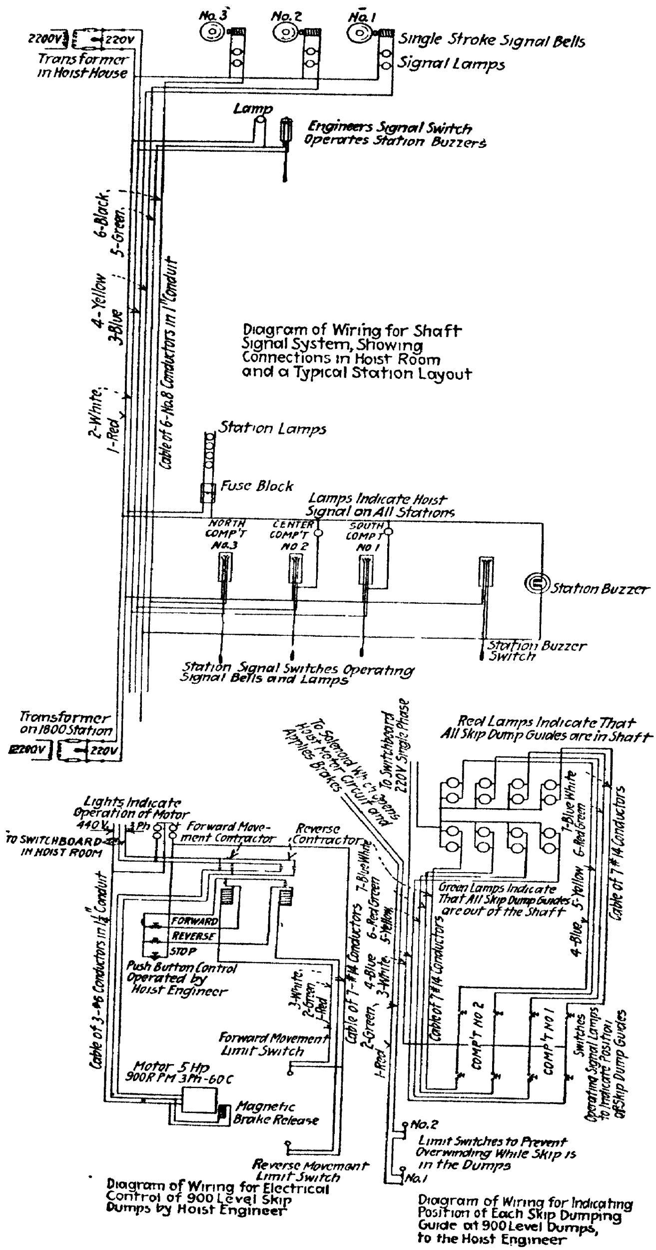


FIG. 8.—DIAGRAM OF WIRING 900-FT. LEVEL DUMPS.

standing. From the top of these pockets, chutes were run to the skip-dump at the 900-ft. level; these are 5 ft. square in section and are untimbered.

900-ft. Level Dump

The skip movement in dumping was worked out graphically before the curvature of the guides was decided upon. These, as shown in Fig. 7, are bent on three simple curves and are very satisfactory in operation. As this dump is located at an intermediate level of the shaft and as it is desirable to hoist to the levels above this and to the surface, the dump-guides must be moved out of the shaft at times.

To accomplish this, the guides are broken and the lower portion is hinged. The upper portion is riveted to a base plate which is securely bolted to the shaft and dump timbers. The lower portion is riveted to three bearing plates and is hinged so that a short movement will make it clear the shaft and allow the skip to pass. The rivets are countersunk in the underside of these bearing plates and make a smooth contact with the base plate, which is securely bolted to the shaft and dump timbers. Two pieces of 16-lb. rail, bent to the proper radius and riveted to the roller guides, run in slots cut in the base plate and timber and guide the movement of the dump-guides as they are moved in or out of the shaft.

The four dump guides, two for each skip, are moved simultaneously. Each guide is connected by two links to a shaft keyed to a wormwheel, which is connected by gearing to a 5-hp. motor. To get the necessary movement, about 20 in., requires about one-fifth of a revolution of this wormwheel, which takes about 11 sec. The motor is rated at 720 r.p.m. and the total reduction of speed through pinion, gear, worm, and wormwheel is 684 to 1.

The motor is governed from the hoist-operating platform by a push-button control and solenoid-operated contactors; see diagram in Fig. 8. When the limit of movement of the dump guides is reached, in either direction, the circuit is automatically broken by limit switches, which are operated by a trip arm attached to the wormwheel. The motor, being equipped with a magnetic release brake, is stopped immediately when the circuit is broken.

The small oil switch on the hoist switchboard has a potential release coil which opens at low voltage. It is also opened by a time-limit relay, which operates in case the current flows for a longer time than necessary. The four sets of links, operating the dump guides, are equipped with contacts which light lamps on the hoist-operating platform, thus indicating to the operator that each guide is in either dumping or non-dumping position. Another bank of lamps indicates whether or not the motor is operating. The operator may stop the motor by an emergency push button in case the automatic limit switches fail to act.

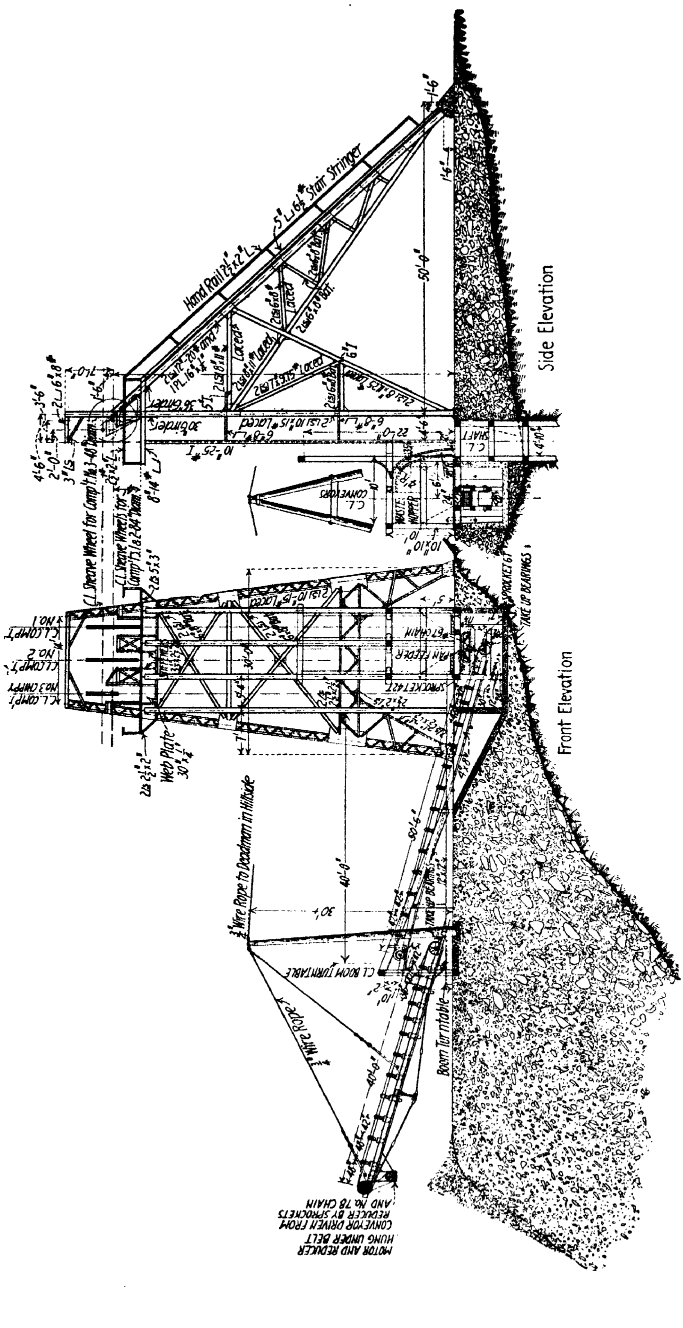


FIG. 9.—STEEL HEADFRAME, SKIP DUMP, HOPPER AND CONVEYORS FOR WASTE DISPOSAL AT PARVENU SHAFT.

To prevent overwinding while the skip is dumping, limit switches are used. These are operated by the skip rollers at a predetermined point along the roller guides. They break the emergency solenoid circuit, causing this device to act as above described. The wiring diagrams are given in Figs. 5 and 8.

To divert the ore from one hoisting compartment to the chute opposite the other hoisting compartment, the diversion truck shown in Fig. 7 was designed. It is operated by hand and when it stands in the position shown, the ore will go directly from the shaft into its regular chute; but when it is pushed to one side or the other, the flow is diverted as desired.

Surface Dump and Waste Conveyors

Figs. 9 and 10 illustrate the surface dump and method of handling waste. A belt conveyor transports the waste from the hopper at the skip dump to the edge of the waste pile. Due to the steep slope of the

Fig. 10.—View of head frame and swinging boom conveyor.

hillside at the shaft collar, the waste dump is inclined to creep, especially during the spring thaw. Therefore, a continuous stationary conveyor would be impractical; it would be difficult, if not impossible, to keep it in alignment without undue attention and expense. To overcome this difficulty, two conveyors were used. The first, which takes its feed from a pan feeder under the skip-dump hopper, is of the stationary type and was extended as far from the shaft as was deemed safe, about 40 ft. (12 m.). The main members of the framework for this conveyor are anchored to the concrete walls of the pit, so, even should the dump move

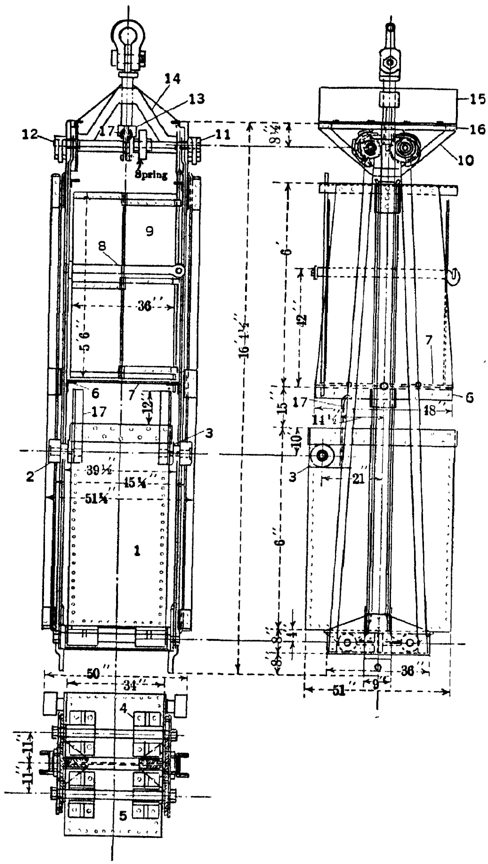


FIG. 11.—SKIP AND CAGE.

1. SKIP BOX. 2. SKIP-BOX ROLLER. 3. ROLLER SUPPORT. 4. SKIP-BOX BEARINGS. 5. SKIP-BOX REST. 6. CAGE-FLOOR SUPPORT. 7. CAGE FLOOR. 8. CAGE BAR. 9. CAGE DOORS. 10. CAGE-BONNET SUPPORTS. 11. DOGS. 12. DOG SHAFT. 13. CHAIN LINK. 14. CROSS HEAD. 15. CAGE BONNET. 16. CAGE-BONNET SUPPORT. 17. SKIP HORNS.

from under it, this conveyor would be practically self supporting. The second conveyor is of the swinging-boom type. The framework is all steel and is suspended from a bipod. The lower end is hinged on especially designed bearings on a turntable under the head pulley of the first conveyor. This construction not only makes the conveyors independent of the sliding dump, but also, as the boom may swing in an arc of about 90°, the total capacity of the piling system is greatly increased.

The conveyors are operated with 5-hp., 440-volt, three-phase, 60-cycle, induction motors, rated at 1700 r.p.m. D. O. J. speed reducers are used, the ratio of reduction being 1700 to 25. The belt speed of the conveyors is 155 ft. per min. The motors are controlled from the operating platform of the hoist.

The Head Frame

The head frame contains no particularly new features. It is a sturdy well-built affair, but is not quite high enough for our present purposes. An additional 10 or 15 ft. clearance over the skip when it is in dumping position would be desirable.

Skip and Cage

The skip and cage are combined. On account of the low head frame it is necessary to put the cage floor directly over the skip box, leaving only enough room for the latter to work freely. This arrangement would interfere with loading; therefore the floor was made in four sections, hinged so that it can be folded back when rock is being hoisted. The floor is of $\frac{3}{8}$-in. (9.5-mm.) plate and is supported by 4 by 4-in. angles (10 by 10- m.), which are bolted to the bale and braces. When men are being hoisted or lowered, doors are used.

The cradle, head rigging, and safety devices contain no particularly new features. The latter are of the usual dog type, actuated by the release of a chain wheel and spring. The cradle consists of a piece of 8-in. (20-cm.) I beam and two pieces of $2\frac{15}{16}$-in. (7.5-cm.) shafting secured, by cast steel brackets, to the cradle side plates and bales. The bales, or main side members of the frame are of $4\frac{1}{2}$ by $1\frac{1}{2}$-in. (11.4 by 3.8-cm.) mild steel, the width at the bottom is increased to 9 inches.

The skip-box sides are of $\frac{1}{4}$-in. (6.3-mm.) plate with $\frac{3}{16}$-in. wearing plates in front and back. The bottom is of $\frac{3}{8}$-in. (9.5-mm.) plate with a $\frac{3}{16}$-in. wearing plate on top of a false bottom of 2-in. (5-cm.) plank. The capacity of the box is about 70 cu. ft., or $4\frac{1}{2}$ tons of ore. The total weight of the skip and cage is about 5000 pounds.

Signal System

The electric signal system for the operation of the skips is a development of the multiple-bell circuit; the wiring diagram is given in Fig. 8.

Two 220-volt mains and four commons extend the entire length of the shaft. These are No. 8 wire, in a 6-wire cable, which is run in conduit and terminates in weatherproof junction boxes on each station. There are four complete circuits, one for each of the two main hoisting compartments and one for the "chippy cage" compartment; the fourth is for the station buzzers, which are used to call the cage riders.

The Anaconda mine post switch is used. This is a cast-iron, bottle-shaped affair and contact is made by pulling a rope connected to the contact points inside of the bottle. These points work against the action of a spring. The bells are of the solenoid-operated, single-stroke type.

Costs of Installation

The following is a statement of the costs of the shaft and equipment.

Year		Feet		Per Foot
1909	Raising shaft from 1000-ft. to 700-ft. level..	321.5	$ 9,800	$30 48
1913	Sinking shaft from 1000-ft. to 1150-ft. level..	152.3	10,300	67.62
1915	Sinking shaft from 1150-ft. to 1300-ft. level.	148.6	11,300	76.04
1916	Sinking shaft from 1300-ft. to 1500-ft. level...	200.7	19,000	94.66
1916	Raising shaft from 700-ft. level to surface..	726 0	23,000	31.68
1917	Sinking shaft from 1500-ft. to 1800-ft. level and to sump........	400.0	32,700	81.72
		1949 1	$106,100	$54.44

1916	Hoist and equipment, house, foundation, motor, electrical equipment, etc.....	$45,000	
	Head frame, foundation, etc	15,000	$ 60,000
1919	Conveyors, skip dumps, hopper, etc. at the surface.......	5,000	
	100-ft. level pocket and equipment........	3,900	
	1000-ft. level pockets and 900-ft. level skip dumps and equipment....	18,000	
	1300-ft. level pocket and equipment	6,200	
	1500-ft. level pocket and equipment..	6,200	
	1800-ft. level pocket and equipment	7,600	
	Skips......................	3,100	50,000
			$216,100

Conclusion

In conclusion, it might be said that no attempt has been made to go into the great mass of detail involved in constructions of this nature. Only the important features have been touched. Each mine has its own peculiar problems and it is impossible to prescribe a standard solution for all.

This paper has been written with the permission and approval of V. S. Rood, Manager.

DISCUSSION

E. M. Norris,* Butte, Mont. (written discussion).—It is well to keep in mind the limitations of the type of hoist described. Its control is largely dependent on the operation of the brakes; though the engine can be plugged (motor reversed while up to speed) without apparent injury to the electrical equipment, this is considered bad practice. If the electric power is cut off suddenly, the control of the hoist is altogether dependent on the brakes; this control appears to be satisfactory with medium loads at moderate hoisting speeds.

Two electric hoists in the Butte district are similar in design to that installed at the Parvenu shaft. They are economical in operation and respond more quickly and smoothly to the application of motive power than do hoists operated by steam or compressed air. They hoist from a depth of 1000 ft. (304 m.) with a maximum load of six tons; their capacity appears to be limited to 900 tons of material handled per 16 hr. of hoisting time.

* Assistant General Superintendent of Mines, Anaconda Copper Mining Co.

Octagonal Ventilation Shaft of Davis-Daly Copper Company

BY J. L. BRUCE, BUTTE, MONT.

(Wilkes-Barre Meeting, September, 1921)

FOR a number of years, from an economic standpoint, the ventilation of the Colorado mine of the Davis-Daly Copper Co. has been a difficult problem. The development through the main hoisting shaft was pursued for a number of years and to a depth of about 2500 ft. (762 m.) before large bodies of good commercial ore were opened. This lack of early success in exploration restricted expenditures for ventilation, with the result that the present orebodies were developed at a depth of 2500 ft., and in country rock of relatively high temperature, without adequate means of supplying the development faces and the stopes with cool or fresh air of low humidity and suitable velocity. During the progress of development, two or three connections were made at higher levels with the workings of adjoining operations, but these were of a makeshift and unsatisfactory character and afforded no means for carrying downcast air from the surface, in anything approaching a direct course, to the working faces.

During the latter part of 1917, arrangements were made with the Anaconda Copper Mining Co., whereby, during the year 1918 and the early part of the year 1919, a crosscut was driven eastwards on the 2500-ft. level of the Colorado shaft and connection was made by means of an up-raise of approximately 200 ft. to a crosscut driven westwards from the 2800-ft. level of the Belmont mine, this crosscut at the top of the raise being driven at Davis-Daly expense by the Anaconda Copper Mining Co. This connection, in conjunction with a No. 11 Sirocco fan installed near the Colorado shaft, provided the Colorado mine with a maximum of approximately 30,000 cu. ft. per min. of relatively fresh air, most of which came in a fairly direct route from the downcast Belmont shaft through the No. 11 fan to the 2500-ft. level stopes. After passing through these stopes and the stopes of overlying levels, it passed through the main crosscuts on higher levels back to the Colorado shaft, which provided the upcast for this current of air. The improvement noted on the completion of this work was marked, and, for a time, fairly satisfactory. Further extension of the mine workings, however, and

the exposure of more faces, and increased rock and water temperatures in the development of the 2700-ft. level, early in the year 1920 caused the ventilation conditions of the workings again to become very unsatisfactory. It was found that many faces in the mine, even when afforded the best local ventilation practicable, recorded temperatures of from 90° to 100°, with humidities as high as 100 per cent., even when many working faces were ventilated with compressed air furnished by three compressors, which were kept operating almost constantly, at an expense of about $50 per day above that normally required for rock drilling and other mechanical operations.

Means were considered for again improving the ventilation and affording a plan whereby this could be extended downwards and outwards as the mine became deeper and more extensive in its developments. In planning for this the following principal factors were considered:

1. *Ventilation to be Independent of Adjoining Properties.*—This was considered highly essential, particularly on account of the fact that air coming from adjoining workings had at times contained noxious gases from underground mine fires. The occurrence of such fires or gases in adjoining workings, if these workings were depended on for the supply of fresh air, would be likely to compel the discontinuance of operations at the Colorado mine. Furthermore, on occasions, it had been found that the supply of air from adjoining workings had been so cut off or restricted that the quantities to be depended on were altogether uncertain. Also it was believed that the ventilation of the mine should be independent of other properties so that in case of fire in the Colorado mine the reversal of ventilation currents and distribution of air would be under the direct control of the management.

2. *Downcast Operating Shaft.*—It was decided that many benefits would be obtained by making the main operating shaft downcast instead of upcast. The upcast air was known to contain entrained copper water, which, though small in quantity, was sufficient, at the high temperatures of the upcast air, to cause serious deterioration of air lines, pump columns, and particularly hoisting ropes, skips, and cages. An upcast main hoisting shaft maintains pump stations, shaft stations, and usually the principal crosscuts, underground stables, etc. in a hot, humid, and uncomfortable condition; this makes local ventilation by small fans or by blowing of compressed air a necessity, whereas a downcast main hoisting shaft maintains nearly all such places in a cool and dry condition.

3. *Distribution of Air.*—By so locating both the main downcast course and the main upcast that each is connected with practically every level in the mine, it is possible in most cases to split off the desired amount of fresh air at any desired level, either by short-circuiting to the upcast

course or by auxiliary fans, so that the fresh air will travel by the most direct route to the places at which ventilation is necessary, and after being used will go as directly as possible to the main upcast course.

4. *Location.*—The location of the ventilation shaft was chosen particularly with a view to the rapid and economic completion of the work. It was preferable to locate the surface equipment upon surface property already owned by the Davis-Daly Copper Co., inasmuch as the value of all other surface property suitably located would have been high. Location close to the hoisting shaft was also desirable, from the standpoint of convenience of inspection of the surface fan and equipment and economy of operation. A point about 110 ft. (33 m.) to the southwest of the Colorado shaft was chosen, as the ground was known to be reasonably free from geologic structures, such as faults and veins, that would indicate bad ground. The exact point for location of the center of the shaft was then selected by trial, as being the point that would require the minimum total length of crosscuts from the various existing levels, while leaving a suitable amount of space for the plant on the surface.

5. *Cross-section of Ventilating Shaft.*—The original plan contemplated the construction of a standard size two-compartment shaft with each compartment 4½ by 5 ft. in the clear. It was intended, however, to make this shaft smooth-lined by cribbing solid with wall plates, end plates, and centerpieces set tightly face to face, leaving out the posts entirely. The suggestion was made by A. S. Richardson, of the Anaconda Copper Mining Co., who was engaged on the work as consulting ventilation engineer, that the cross-section could be much reduced, without sacrificing air capacity, by the construction of a circular, reinforced concrete-lined shaft. While this was considered impracticable, both from the standpoint of estimated cost and time of completion, the idea was evolved of building a smooth-lined shaft of octagonal cross-section. It was determined that an octagon circumscribed about a circle with a diameter of 6½ ft. (3.3 m.) and with an area of 34.5 sq. ft. (3.2 sq. m.) in the clear, would give at least as great a capacity as the two-compartment shaft referred to, with an area of 45 sq. ft. (4.1 sq. m.) in the clear. This was based on experiments made by A. S. Richardson, who estimated, on the basis of resistance tests made on miniature air passages, that the coefficient of friction in the octagonal, smooth-lined shaft would be about 0.000,000,001,35. As a matter of fact, the tests made after completion of the work showed a coefficient slightly less than this.

Based upon experience gained in the use of cribbed shaft timbering at the Butte & Superior mine, the intention had been to use shaft timbering or lining 10 in. (25.4 cm.) thick for the standard two-compartment shaft. Inasmuch as the span of timber for the octagonal shaft

would be less than 3 ft. (0.9 m.) as compared with 4.5 to 5.0 ft. for the rectangular, and as the size of excavation would be approximately one-third less and the natural arch of the rock better on account of the circular shape, it was decided that the timbering or lining of the octagon need be no more than 8 in. thick to give the same strength of support. It was found, by calculation, that the amount of timber saved by using the octagonal cross-section would be about 42 per cent. of that required by the rectangular; or in other terms that the rectangular shaft would have required 50 per cent. more excavation and 72 per cent. more timber than was actually used. As the work of handling the waste and timber in the main hoisting shaft was a serious obstacle to the maintenance of normal production, this was a highly important advantage.

History of Progress

The proposal to sink a new ventilation shaft, and the general plans thereof, were submitted to the Board of Directors on Aug. 14, 1920. These were approved and preparatory work commenced on Sept. 3. Re-surveys of all levels down to the bottom of the proposed shaft were made, by carrying meridian lines down the main shaft by means of plumb wires hung about 3 ft. (0.9 m.) apart and carried down not more than 400 ft. for each plumbing. These were rechecked until meridians agreed within a limit of 3 min., which was considered satisfactory, in view of the strong air current in the shaft and the short base lines. This opinion was confirmed by the raise connections, none of which was in error as much as 3 inches.

Before starting crosscuts to reach the site of the new shaft, it was necessary to clean out and retimber old shaft stations, lay turn sheets and tracking, and install lights and signals. Three places were started on Sept. 3, two on Sept. 9, three on Sept. 13, and one on Sept. 24. By the end of September there had been completed 325 ft. (99 m.) of crosscuts and 30 ft. (9 m.) of raising, including as such the new shaft station sets. Table 1 shows the rate of progress and its location for each month. All of the shaft work was done by raising, with the exception of 90 ft., which was sunk from the surface after Dec. 1 to hole the raise from the 300-ft. level, and 12 ft. of winze sunk from the 1500-ft. level after Feb. 4, 1921, which holed through to the raise from the 1700-ft. level, completing the driving of the shaft on Feb. 16. Between this date and the starting of the exhaust fan on March 21, all chute timber, ladders, slides, etc. were removed, station sets cribbed, crosscuts cleaned, two tight doors constructed in each crosscut, drain pipes placed, and other things were made ready.

TABLE 1.—*Rate of Progress Each Month*

	Level									
	300	400	600	800	1000	1200	1400	1500	1700	Total
	Advance in Crosscuts, in feet									
September, 1920	62	28	56	4	59	6	30	30	50	325
October	0	42	35	96	0	6	0	38	60	277
November										
December										
January, 1921										
February 1–20										
Totals	62	70	91	100	59	12	30	68	110	602
	Advances in Raises,* in feet									
September, 1920	10	0	0	0	10	0	10	0	0	30
October	66	18	22	10	46	24	42	20	12	260
November	75	48	48	38	30	48	28	41	60	416
December	111	52	82	54	48	78	46	36	74	581
January, 1921	38	42	48	38	66	76	50	29	56	443
February, 1–20	0	0	0	00	0	0	24	0	51	75
Totals	300	160	200	140	200	226	200	126	253	1805

*Stations included as raising total height 90 ft.

TIMBER AND TIMBER FRAMING

In planning the octagonal shaft, it was decided to use 8 by 10 in. lumber, which could be obtained in native fir of good quality at a satisfactory price. It was realized that it would be necessary to cut and frame approximately 17,000 pieces for the shaft lining, in addition to those necessary for stations, chutes, partitions, etc., and that each piece to be framed was only about 3 ft. long, so that any loss of timber in framing would be a relatively large proportion of the amount in each stick. The lumber required for cutting the halved joint customarily used for wall and end plates would have been just 25 per cent. greater than that actually used, so the expense of such framing would have been almost prohibitive. A miter joint would have left a continuous joint at each of the eight corners of the shaft, and would have been unstable and altogether unreliable for distributing the pressures. Eventually the style of framing illustrated in Fig. 1 was developed. This gave at each corner of the shaft a joint which could not easily be displaced.

By reversing the timber in each alternate ring of timber bricking in the shaft, as shown by the broken lines A, the joint at each corner was also broken vertically at 10-in. intervals. This has proved to be entirely satisfactory.

An ideal arrangement for framing this bricking would consist of a roller carriage across two saws, one with the arbor placed at an angle of 45° with the line of travel of the timber and moving vertically to make the cuts marked B, Fig. 2(a), the other with the saw arbor set perpendicular to the line of travel and moving horizontally to make the cuts marked

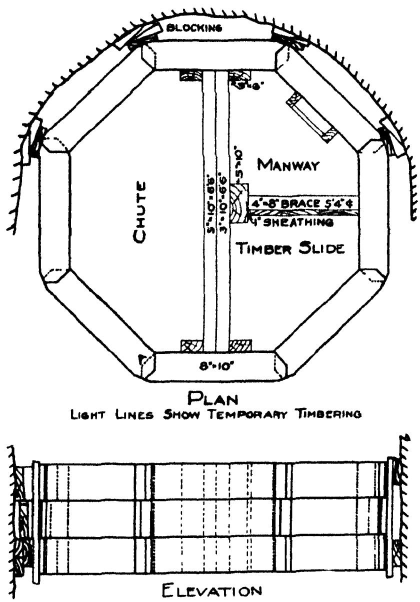


FIG. 1.—DETAILS OF CRIB SET FOR AIR RAISE.

A, the timber being turned over 90° between the two cuts, and each saw being adjusted to limit the depth of the cuts. With this arrangement, the timber, always progressing, would require only two cuts for each piece of bricking, after which cuts C and D could be made on each piece of bricking on a sliding table saw. By this method, there would have been little waste aside from the sawdust, as the block left after cutting five pieces from a 16-ft. timber would have been serviceable for blocking.

As the work was carried out, no special saws nor equipment were installed; use was made of machinery in operation in almost any small mine framing shop. The first operation on a 16-ft. stick of 8 by 10 in. (20 by 25 cm.) timber was the making of cuts A, Fig. 2(b), on the regular

swing cut-off saw. Cuts B were then made on the same saw by shifting the pieces to the required angle with the saw, which was accomplished by fences and stops. This produced five pieces, each of which was taken to a sliding table saw, where cuts C and D were made, of equal length, and with a constant distance between cuts A and C, which was accomplished by suitable fences and stops attached to the saw table. Cut E was then made by placing the stick in a cradle form, secured to the carriage of a Denver timber framer, so that the piece of bricking rested on the cut D, which was horizontal, bringing the piece into position for making cut E with the top cut-off saw of the framer, the horizontal saw

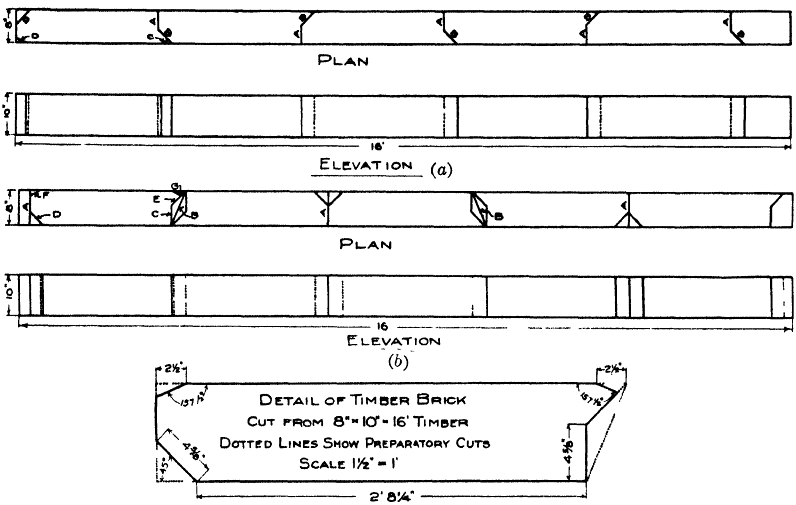


FIG. 2.—DETAILS OF FRAMING FOR SHAFT.

of the framer being disconnected for this operation. Cuts F and G were then made on the sliding table saw by resetting the stops. This sequence of making the cuts throws all the irregularities due to varying dimensions of the 8 by 10 in. timbers to the outside of the shaft lining, and provides a good fit for the joints and a constant size for the exposed side of the lining. The total waste by this method of cutting amounted to only 5 per cent. of the timber purchased for the bricking.

Early in the work of placing the timber in the shaft, it became apparent that the variation in thickness of the rough timbers, if not compensated, would cause the timbering to build higher at one side of the shaft than at another, with the resultant tendency to deflect the walls out of plumb. This could have been corrected by sizing one side and one edge of the 8 in. by 10 in. by 16 ft. timbers before framing, to give uniform thickness; but as this was not convenient it was decided to sort the

bricks into piles of uniform thickness, each raise to take all its bricking from one pile.

The octagon was subdivided into three compartments, as shown in Fig. 1. The largest, consisting of one-half of the shaft, provided a chute for the waste broken, and the other two, which were equal in size, provided the manway and the timber slide. These were separated by horizontal braces 5 ft. 4 in. (1.62 m.) apart, which supported the chute wall. This was constructed of double 3 by 10 in. planking, which, after removal, was used for flooring in the mine stopes. At first, some trouble

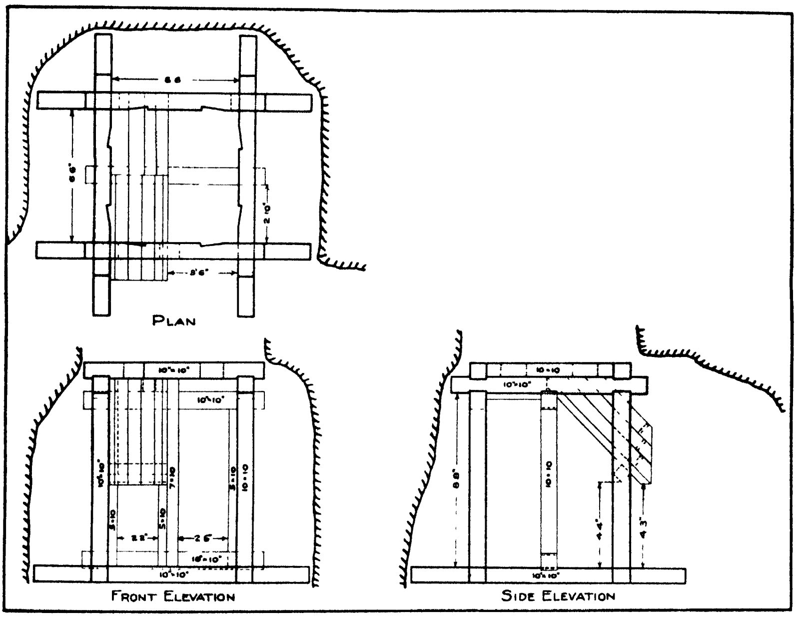


FIG. 3.—STATION SET FOR AIR RAISES.

was experienced by the breaking of this planking from the concussion of shooting or of falling rock. This was overcome by leaving spaces in the chute wall by inserting 2-in. blocks between each pair of planks and the pair above them. The manway and timber slide were separated by 1-in. sheathing nailed to the 4 by 8 in. braces, thus providing a smooth-lined compartment for the timber slide and giving a division for upcast and downcast air currents to improve the ventilation. After completion of the raises, all timber within the octagon was removed and most of it was used elsewhere in the mine. During the raising, no lining was used on the bricking exposed to the wear of the rock in the chute and it was found that in the highest raises there was little wear.

The plan of raising from nine levels made necessary the construction, at each level, of working stations at which the waste would be drawn from a chute at the bottom of the raise and where a small hoist would be installed for hoisting tools, timber, blocking, etc., up the raise, the hoist being set on the opposite side of the crosscut from the raise, so that ample room would be left for egress and ingress to the manway compartment and the timber slide. The first station set framed was erected on the surface and subjected to the inspection and suggestions of engineers, foremen, bosses, contractors, and all concerned. From this coöperative effort, the design shown by Fig. 3 was evolved. All sills were made 14 ft. long, so that the station set would have substantial support when holing the raise from the level beneath. At the conclusion of the raising, all timbers in the station set were removed, with the exception of the sills, the four corner posts, and the square frame on top of the posts. The octagonal section was then carried upwards through the station set by bricking in the openings, leaving a door on the side next the crosscut.

Equipment and Methods Used

Crosscuts were driven with the No. 21 Waugh Turbro and the No. 248 Leyner Ingersoll rock drills, using 1¼ in., round, hollow, drill steel. Raises were driven with the Ingersoll-Rand CC-21 stope hammer, using 1¼ in. cruciform steel. It was customary to drill a four-hole center cut with eight more holes placed one above each corner of the raise and pointed slightly outwards from the center.

The timber hoists were of various types and makes of small air hoists and were located opposite the timber slide and on the opposite side of the crosscut from the chute mouth. The sheave wheel at the top of the raise was supported by the usual extension gin pole, made of telescoped pipe, and was adjustable in length from 10 to 16 ft. A sheet steel cage or box 20 in. (0.5 m.) square and about 5 ft. (1.52 m.) deep, with a bale at the top, was at first used for hoisting eight pieces of bricking or a load of blocking or wedges, but this was found to be too heavy and too large to be handled conveniently at the top of the raise, and the bale prevented its being raised as near the top of the raise as was convenient. After cutting down the height so that the box would take only four pieces of bricking, without securing satisfaction, a lighter box or basket was designed of old belting laced together. Eventually, however, most of the contractors reverted to the use of the common hooks and chains.

In raising, which was done by contract, three men on each shift constituted a crew, two working in the raise and one operating the hoist and doing the tramming when not otherwise occupied. The best success was obtained when breaking rounds of not to exceed 3 ft., and by keeping the timbering close up to the back. At first an attempt was made to block the timbering with blocking a few inches from each end of each

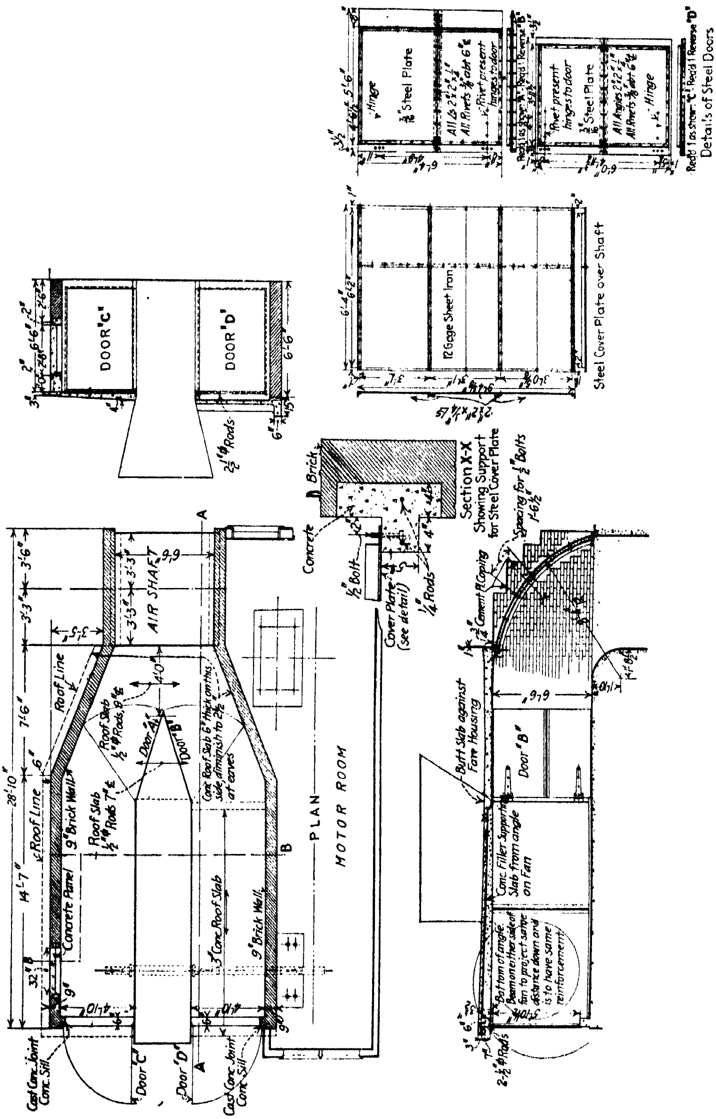

PLAN
MOTOR ROOM
AIR SHAFT
DOOR "C"
DOOR "D"
Section X-X
Showing Support
for Steel Cover Plate
12 Gage Sheet Iron
Steel Cover Plate over Shaft
Details of Steel Doors

FIG. 4.

piece of bricking. This, however, was found to be slow and difficult. As a result the corners of the outer octagon were cut off and after about three complete rings of bricking were placed, boards or planks were set vertically against these at every corner and the blocking completed by gradually tightening with wedges all round the octagon, as shown in Fig. 1. Reasonably uniform and complete distribution of the pressure was assured by the fine rock filling, which was permitted to accumulate between the bricking and the walls of the excavation.

Chutes were kept nearly full, the rock being drawn down just enough before each blasting to make space for the rock to be broken. When blasting, the manway and timber slide were covered, leaving out a sufficient number of the planks at the top of the chute lining to enable the workmen to get through after the blasting.

The plumbing of the timber was checked at intervals of about 100 ft., by the engineers, by means of two plumb lines hung in the manway and timber slide, from a sprag above the timber. A straightedge was then placed, just free of contact with the wires near the bottom of the raise, and two corners located by taking the angles from the straightedge through each wire to such corners with a Stanley adjustable bevel square and by measuring the distances to such corners. These points were then projected to a point near the top of the raise by duplicating the angles and distances on a similar straightedge set in line with the wires just below the top of the timbering.

Doors in Crosscuts

In each crosscut connection between the downcast shaft and the new upcast ventilation shaft, and in other crosscuts in the mine connecting the downcast air course and the upcast, there were installed two air doors, usually from 25 to 50 ft. (7.6 to 15.2 m.) apart. The framework for each door was made by excavating a channel of rock from the bottom, sides, and back of the crosscut, setting two 10 by 10 in. (25 by 25 cm.) posts the requisite distance apart, and separating them at the top and bottom of the drift by spreaders. The forms were then placed between this framework and the rock in place at the outside of the channel and the space filled in with concrete. A door stop inside the 10 by 10 in. framework was made of 3 by 6 in. material and the doors themselves were made of double 1 in. boards with roofing paper between to reduce the leakage of air to the minimum. After completion, all openings were stopped up as completely as possible by calking with oakum, and the doors were held tightly in place by substantial button latches.

On the 300-ft. level, the highest level connecting between the two shafts, small auxiliary trap doors or valves about 18 in. square, opening toward the downcast shaft, were installed. The purpose of these is to assist in the quick reversal of the air current in the downcast shaft

whenever the exhaust fan is reversed, so that in the event of a fire near the collar of the downcast shaft or in the downcast shaft above the 300 ft. level, the quick reversal of current will prevent the smoke from being carried into the underground workings.

Equipment Specifications

Mine Ventilation Fan

It was specified that this fan was to operate at an elevation of 5700 ft. above sea level, and to exhaust 125,000 cu. ft. of air per minute, against a resistance equivalent to a static pressure of 5 in. of water. The fan is of double inlet, reversible type, arranged for belt drive from an electric motor; there are three bearings, one outside of pulley, all supported on cast-iron stands independent of the fan casing and extending down to the floor line. An expansion discharge stack is provided, to reduce the velocity of the air to about 1650 ft. per min. The fan is three-quarter housing and so arranged that no part of the connecting ducts serves as any part of the fan casing or supports the discharge stack. The fan shaft is of best quality hammered steel, about 0.35 per cent carbon, and the fan castings are of tough gray iron free from injurious cold shorts or blowholes.

The bearings are of standard length, double ring oiling, self-aligning, and guaranteed to be sufficiently tight that the suction of the fan will not draw the oil out of them. The fan casing is constructed of ¼ in. steel plate thoroughly stiffened with steel angles and provided with supports for bearings on the foundations. The fan is equipped with reversing damper provided with hooks and eyebolts for locking it securely in position independently of any cable or chain used in hoisting or lowering it. All parts of fan and casing were properly fitted and the wheel balanced before shipment.

Motor Specifications

Fan motor is of squirrel cage type, 600-r.p.m., three-phase, 2200-volt, and with a rating of 200-hp. capacity for a rise of 50° C. Pulley on fan is 40 in. in diameter; pulley on motor, 28 in. in diameter; belt, 16 in. six-ply endless rubber.

General

All woodwork in ore bins, ore sorting room, skip change pocket, etc. in the vicinity of the downcast shaft was thoroughly coated with a fire-resisting paint. The downcast shaft, near its collar, was completely surrounded by a ring or collar of 4 in. piping, perforated with a row of holes so directed that water turned into this pipe will be thoroughly distributed around the shaft timbering. This system is connected with

the water main by a valve on the surface, which may be opened promptly in case of fire in the shaft or near its collar. Inasmuch as there is a heavy condensation of water in the upcast shaft, and as it is smooth lined throughout, it is considered to be practically fireproof.

After starting the fan on March 21, 1921, tests showed that the coefficient of friction in the new shaft was 0.000,000,00129, as compared with the estimate of 0.000,000,00135 made before work was started, and with a coefficient of 0.000,000,0022 for one of the smooth gunite lined, rectangular shafts of the district, and with a coefficient of 0.000,000,0088 for a rectangular, unlined shaft. The volume of air passing was a little more than 90,000 cu. ft. (2527 cu. m.) per min. Under these conditions, the total shaft resistance for 1800 ft. of depth amounts to only 1.15 in. of water, leaving an effective fan suction of about 4 in. of water at the bottom of the shaft, in addition to the natural draft, to overcome the resistance of the remainder of the mine workings. The velocity of the air in the upcast shaft under these conditions is about 2600 lin. ft. per min.

Tests were made at each pair of doors between the two shafts to determine the approximate leakage. This was done by cutting a circular hole 4 in. in diameter in the door nearest the downcast air course and measuring the flow of air through this hole. It had been determined that the hole was large enough to take all the leakage of the other door with practically no resistance by observing that the gage pressures were practically equal on the two sides of the door through which the hole was cut. As the test was made with only one door offering resistance, it could be presumed that the leakage with the holes closed would be substantially less. The test on the various levels showed the leakage to vary from 70 cu. ft. per min. to 600 cu. ft. per min., with a total for the nine levels of 3250 cu. ft., or about 3.6 per cent. of the total air actuated by the fan.

The temperatures in the working places of the mine were greatly improved and the average efficiency of all underground employees was increased at least 50 per cent. Table 2 shows conditions in some typical portions of the mine before and after the work was completed. The temperatures were taken before the air shaft was finished and 24 days after the surface fan was started.

It has been confirmed that the saving in electric power alone, for air compressing, is equal to more than $18,000 per year, when operating the mine under its normal conditions of two shifts in the 24 hr. Only four accidents, involving a loss of time, occurred during the progress of the work, out of 5787 shifts. Two of these were caused by falling ground, one by a rock from the chute mouth, and one by the explosion of a priming cap. None proved serious.

TABLE 2.—*Comparative Temperatures and Humidities at Colorado Mine*

Location	Before			After			Differences		
	Wet Bulb Degrees	Dry Bulb Degrees	Relative Humidity Per Cent.	Wet Bulb Degrees	Dry Bulb Degrees	Relative Humidity Per Cent.	Wet Bulb Degrees	Dry Bulb Degrees	Relative Humidity Per Cent.
2100 station	89	90	96	62	65	85	27	25	11
2336 stope	87	89	92	83	84	96	4	5	+4
2320 drift	89	91	92	88	90	92	1	1	0
2301 northwest at 2311 X cut	95	97	93	81	87	77	14	10	16
2474 rill stope	90	93	89	85	90	81	5	3	8
2421 X cut	89	91.5	92	83	89	78	6	2.5	14
2406 drift	89	93.5	82	86	93	76	3	0.5	6
2450 chute	89	91	92	81	87	77	8	4	15
2568 stope	86	90	85	81	84	88	5	6	+3
2720 int. X cut	92.5	93	98	79	81	91	13.5	12	7
2720 stope	90	93	89	79	82	87	11	11	2
2701 X cut at switch	89	90.5	92	76	80	83	13	10.5	9
2702 northwest drift	84	89	81	74	79	79	10	10	2
2700 station	88	88	100	68	72	81	20	16	19

TABLE 3.—*Ventilation Development Cost*

	Total Advance	Explosives	Compressed Air	Rock Drill Repairs	Steel Consumption	Steel Sharpening	Timber	All Other Supplies	Total Supplies	Total Labor	Total Cost
Preparatory								$ 181 11	$ 181.11	$ 1,281.91	$ 1,463.02
Crosscuts.	602 ft.	$1,429.28	$ 538 32	$ 453 44	$133.97	$ 123 37		426.00	$ 3,104.38	5,015 37	8,119.75
Stations.	18,063 cu ft.	397 67	171.60	144 34	78.45	49 68	$ 1,584 23		2,425 97	4,467.13	6,893.10
Shaft.....	1,715 ft.	3,992 36	1,470.60	1,185.52	662.00	478.75	18,998 35	567 83	27,355 41	25,318.53	52,673.94
Doors, drains, etc								483 65	483 65	1,587.31	2,070.96
Totals........		$5,819 31	$2,180 52	$1,783 30	$874.42	$651 80	$20,582 58	$1,658 09	$33,550.02	$37,670 25	$71,220.27

Unit Costs

		Explosives	Compressed Air	Rock Drill Repairs	Steel Consumption	Steel Sharpening	Timber	All Other Supplies	Total Supplies	Total Labor	Total Cost
Crosscuts, per ft.....		$2 37	$0.89	$0.75	$0 22	$0 20		$0 72	$ 5.15	$ 8.33	$13.48
Stations, per cu. ft.		0 022	0 009	0 0079	0.0043	0.00275	$ 0 0877		0 134	0.247	0.381
Shaft, per ft......		2 327	0 857	0 691	0.386	0 279	11 077	0 331	15.95	14.76	30.71

Costs

The costs of the work are shown in Table 3. Most of the timber used was native fir, costing $37 per 1000 board feet. The cost of timber shown in the statement includes all material reclaimed after completion of the raise. The cost of explosives used averaged a little more than 20 c. per lb. The standard wage scale for miners during the progress of the work was $5.75 per shift. The earnings on contracts averaged $6.28 per shift. The item "all other supplies" includes the pipe and tracking, though practically all of this was reclaimed after completion of the work.

The total depth of the shaft is 1805 ft. (550 m.), 90 ft. (27 m.) being included in the shaft stations, of which there are nine, each 10 ft. high. The amount constructed by raising was 1607 ft., and by sinking, 108 ft.

The labor cost for the shaft work includes a charge of approximately $3 per ft. for extra work in catching up caving or heavy ground, and for trammers paid days' pay outside of the contracts. It also includes approximately $1 per ft. for tearing out chutes, chute linings, and for temporary repair work.

The total rock excavated from stations and raises amounted to about 127,860 cu. ft., and that removed from crosscuts to about 22,000 cu. ft., estimated altogether at about 12,500 tons.

The details of the cost of timber in the shaft are as follows:

	Per Foot of Raise
Cost of timber for bricking (native fir at $37 per M)	$ 7 48
Cost of framing and delivery to shaft collar (framers $6 per day)	0.76
Cost of temporary timber for chutes, etc. (reclaimable-salvage value about 50 per cent.	1.47
Cost of cutting timber for chutes, etc	0 37
Cost of blocks, wedges, etc.	0.99
Total for bricked portions of shaft	$11.07

Cost of timber for stations	$ 83.80 per station
Cost of timber for cribbing stations after completion of work	63.70 per station
Doors for stations	4.40 per station
Total	$151.90 per station

The costs just given do not include the cost of hoisting the waste or disposing of it after it reached the surface. In most mining operations, the waste could be used for filling stopes, which expense would be taken care of by the value of the waste for such filling. In this instance, it was necessary to hoist all of the waste to the surface and to dispose of it by hauling with motor trucks.

DISCUSSION

D. HARRINGTON,* Denver, Colo. (written discussion).—I have spent the greater part of three years studying ventilation in Butte mines and was in the Davis-Daly property before it was ventilated and when it was partly ventilated, and was doing some work there when the new ventilation installation was placed in operation.

From a critical point of view, it would appear to be poor policy to have the two shafts so close to each other (110 ft.) though possibly this could not be avoided; it would appear also that the cross-section of the new shaft (area 34.5 sq. ft.) is somewhat small and apparently no provision has been made for periodical inspection and possible repair of the shaft.

The effects of the installation were numerous and important and almost immediate. Mr. Bruce stated that over $18,000 would be saved annually in electrical power for compressed air alone and said his underground labor had increased efficiency over 50 per cent. This is attested in a statement on page 374 of the September 10 issue of the *Mining & Scientific Press*, which gives the net profit of the Davis-Daly Co. for the quarter ending June 30 (the first quarter after the ventilation plant operated) as $85,597 against $39,206 in the first quarter, before the installation was in operation. The statement also says that "the number of tons hoisted per shift was probably the highest in the history of the property with operating costs low."

Table No. 2 shows a remarkable decrease in the temperature and humidity of many of the working places in the mine 24 days after the fan started in March. These decreases will undoubtedly be more marked after the fan has been operating some months and especially after the cool air of the winter months will have had an opportunity to exert its influence. In conversation with some of the miners of the Davis-Daly property in Butte in September, 1921, the writer was informed that in some of the previous hot places, the worker now instead of being almost naked uses extra shirts and must "keep the muck stick moving to keep warm."

One of the remarkable facts brought out is that the frictional coefficient of the comparatively smooth octagonal shaft was about half of the similar coefficient for a smooth-lined gunited rectangular shaft and about one-seventh of the coefficient for the ordinary rectangular timbered shaft. This shows that smooth-lining air shafts and adopting circular or octagonal cross-section, at least for shafts used solely for air transmission, greatly reduce the cost of ventilation.

One of the essential and novel features of the installation is the quickness with which it was conceived and the promptness and dispatch with which it was put into effect. The use of the octagonal section, to take advantage of arching of the rock and of minimum rubbing surface for the

* Supervising Mining Engineer, U. S. Bureau of Mines.

flowing air, and of 8 by 10 in. timbers skin to skin, giving a smooth surface also to decrease air friction and to give added strength and stability to the shaft, the installation of a fireproof-housed fan at the surface capable of causing reversal of currents with minimum delay (it has recently been done in less than 10 min.), the care taken to prevent leakage through doors, and the fact that a metal mining company would feel justified in spending over $70,000 on a purely ventilating installation, make the project one of the most interesting examples of later-day progress in metal-mine practice in my observation.

It is to be hoped that the distinct advance made by the Davis-Daly Co. will be followed by others; viz., dividing the mine into separate ventilation splits and keeping them separate; making adequate provision to force moving air by auxilliary ventilating equipment (brattice cloth, small fans, tubing, etc.) to blind ends and in fact to all places where men work.

A. S. Richardson,* Butte, Mont. (written discussion).—The octagonal ventilation shaft of the Davis-Daly Copper Co. is the first attempt in the Butte district so to design an air shaft as to secure the required air-carrying capacity with a minimum amount of excavation and economy of the power required to overcome the resistance to the flow of air. Other shafts in the district were not sunk for such service, being used first for hoisting ore; they, therefore, conformed to the conventional rectangular type.

Although circular shafts have been in general use in Europe and South Africa, they have not been adopted in this country, even in the eastern coal fields, where similar conditions exist. Except for carrying air, it is questionable whether there would be great advantage in the adoption of this type of shaft by deep metal mines; but when the shaft is to be used solely for ventilation purposes, this type is by far the best. All of the advantages of the smooth-surfaced circular shaft are closely realized.

The tests made to determine the leakage through pairs of ventilation doors, described on page 264, are open to objection. Gage readings of the pressure difference between two sides of a door could not determine that there was no leakage through openings other than that purposely left, for 0.01 in. of water would indicate a pressure sufficient to induce a velocity of flow of approximately 420 ft. per min., and this cannot be detected on an ordinary water gage. As there was a flow of air through the hole at which measurement was made, a pressure difference, however slight, must have existed, and there would have been at least some flow of air through all other cracks and openings so that an absolutely exact total measurement could not be made.

The saving of $18,000 per year in power used for compressing air is of importance. Opening the compressed-air line is the most common

* Ventilation Engineer, Anaconda Copper Mining Co.

method of relieving oppressive ventilation conditions; its cost is not generally appreciated, and the figures are of great value for that reason.

An average 50 per cent. increase in efficiency of all underground employees is impressive. Full benefit from improvement in ventilation conditions is seldom realized in increased labor efficiency because standards as to what constitutes a shift's work are clearly defined in the minds of many miners, and these are not easily changed. An exception to this condition occurs where the contract system is employed.

The record of improvement within 24 days after starting the surface fan is good. The experience in other Butte mines has been that the full benefit to be derived from any extensive change in a ventilation system is not experienced until at least one year has elapsed. Table 2 does not show the full significance of the improvement made. The record for stope 2336 shows an increase in relative humidity; this might be misleading unless it is realized that an increase in relative humidity at lower temperatures does not necessarily mean an increase in the moisture content of the air. In the stope under consideration it decreased about 12 per cent. Increase in velocity of air movement, which is one of the most important factors in developing a cooling effect, is not brought out in the comparison, mainly because the instruments in common use are not adapted to the measurement of such low velocities as are ordinarily found within the workings of most metal mines.

Estimation of ventilation conditions, as affected by temperature, humidity, and air movement, cannot be accomplished by any of the methods used in this country. The "Katathermometer," which has been developed in South Africa, seems to be the best effort in this direction because it depends on the rate of cooling of artificially preheated dry and wet bulb thermometers, but more information is needed in regard to its use.

J. L. Bruce (author's reply to discussion).—The paper states quite fully the reasons for locating the ventilation shaft so near to the existing shaft. The controlling factors consisted principally in the rapidity with which the program could be completed, the considerable saving in cost of construction, and the availability of surface space at moderate cost as compared with the location of a shaft closer to the principal underground activities. Except for the resistance of air traveling the extra distance from stoping operations to the foot of the new ventilation shaft, we have found no objection to the proximity of the ventilation shaft to the operating shaft. There is apparently no admixture of the foul gases from the upcast shaft with the air passing into the downcast shaft, and the leakage of air between shafts underground is, in our opinion, not appreciably greater than it would have been were the shafts located 1000 ft. apart.

The ventilation shaft has now been completed for approximately one year, and although some heavy ground was encountered in places, there has been no evidence of any failure or weakness in the shaft timbering. Inspection of the shaft will be accomplished, whenever desired, by erection of a temporary tripod or headframe immediately over the surface opening, from which a special cage or deck will be operated through the shaft by a portable hoist, which will be moved to a suitable location on the surface.

The conclusion has been reached that the diameter of the shaft might have been increased about 12 in., increasing the area approximately 33 per cent., without an increase in the size of the timber used. But as this was doubtful before the character of the ground was known, the smaller diameter was chosen on the presumption that this shaft would always have economic capacity for exhausting a considerably larger volume of air than there would be likelihood of receiving into the mine through the operating shaft in combination with the existing and probable connections to adjoining mines.

Because of operating conditions since April 1, 1921, owing to curtailment of production, no reliable comparison of results could be made, but sufficient is disclosed to show that the estimate of 50 per cent. improvement in efficiency of underground labor is quite conservative.

It has been necessary to continue the operation of auxiliary underground ventilating equipment, including smaller fans to force air from the more important air courses to the dead ends and less accessible portions of the mine through ventilating tubing, but this has been made much more efficient owing to the accessibility of a suitable source of fresh air for these requirements.

While the method of measuring the leakage of air through pairs of ventilation doors, as pointed out by Mr. Richardson, is not extremely accurate, the total of such error would be a very small percentage of the total flow of air through the air shaft.

Dust-ventilation Studies in Metal Mines*

By D. Harrington,† E. M., Denver, Colo.

(New York Meeting, February, 1921)

One of the main functions of the United States Bureau of Mines is to obtain and disseminate information that will promote safety in and around mines, and the health and safety of employees engaged in mining. Serious health and safety conditions arising out of contamination of air in metal mines by dust, fumes from explosives and fires, and absorption of oxygen by timber and other causes, led the Bureau of Mines to coöperate with the United States Public Health Service in the study of dust occurrence and ventilation in metal mines in various parts of the country.

A. J. Lanza, of the U. S. Public Health Service, and Edwin Higgins, mining engineer of the U. S. Bureau of Mines, in 1915, made a detailed study in the mines around Joplin, Mo.; their reports[1] are now available. In 1916, 1917, and 1918, a similar study was made in the Butte, Mont., region by Doctor Lanza and D. Harrington, mining engineer of the Bureau of Mines; the reports of this investigation will be available soon. In 1919, the study was continued in some of the Arizona mines by D. Harrington, of the Bureau of Mines, and G. E. McElroy and R. A. Koronski, of the U. S. Public Health Service, the underground work being completed in June, 1920; these reports are now being prepared for publication. In the spring of 1920, the scope of the work was extended, with D. Harrington in charge as supervising mining engineer, and short studies were made in the mines of the Coeur d' Alenes in Idaho and around Oatman, Ariz. It is hoped soon to extend the work into mines of California, Nevada, Colorado, Michigan, and Minnesota, and ultimately to other states should the mining companies be willing to coöperate.

While the coöperation of the mining companies is necessary, the companies are put to practically no expense other than supplying a guide to accompany the investigators underground, the length of time spent

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† Supervising Mining Engineer, U. S. Bureau of Mines.

[1] Bureau of Mines *Tech. Paper* 105 and *Bull.* 132, and U. S. Public Health Service *Bull.* 85.

underground generally being a few weeks for a comparatively extensive mine.

Tentative Conclusions and Recommendations as to Dust

Investigations in mines of foreign countries, as well as of the United States, prove that breathing air containing minutely divided dusts is likely to be harmful to the health. The most harmful dust is apparently that which is practically insoluble and breaks into minute particles with sharp, cutting edges. Free silica, such as quartz, flint, etc., has these harmful properties to a great degree and is thought to be the most harmful of dusts. Silicates are also insoluble and some of them break into fine particles with cutting edges, but they do not appear to be as harmful as quartz. The harmful dust, in general, is smaller than 10 microns, or about 1/2500 in. Inasmuch as probably much over 50 per cent. of all metal mining in the United States is in siliceous material, the importance of the dust problem is manifest.

While fine siliceous dust is likely to be dangerous, especially when the dust is free, such as quartz, a mine working in quartz material may have especially dusty conditions and the employees be apparently immune to diseases of the respiratory tract for the following reasons:

1. The quartz may be in a condition of partial alteration, due to heat, pressure, or chemical solutions, and bear somewhat the same relationship to unaltered quartz, as to sharpness and hardness, as would decayed oak to the original dry hard material.

2. The quartz dust may be associated with some other dust so that the harmful features of quartz dust (sharpness and insolubility) are neutralized or eliminated. The latter is essentially the adsorption theory advanced by Haldane.

3. A mine having dry siliceous material drilled dry, even using a large percentage of upper holes and hence producing the maximum amount of fine dust, may be comparatively harmless to the health of workers if sufficient air currents are provided to sweep the dust away as it is formed, the workers keeping as much as possible to the fresh-air side of the drill. In general, compressed-air blowers (supplying about 100 cu. ft. of air per min.) are inadequate to ventilate these dusty places properly, so definite means, such as small fans and tubing, must be provided to cause a current of 1000 or more cu. ft. (28 cu. m.) per min. to pass the worker and either take away the dangerous fine dust from the working place or reduce the dust content of the air at the working place below the danger point.

While siliceous dust is probably the most harmful, it is the opinion of the writer that all dusts, especially those insoluble in water (and some that are soluble) are likely to be ultimately harmful to the health of breathers; this applies to coal dust as well as to other mineral dusts.

The writer cannot concur in the recently advocated policy of introducing foreign dusts to neutralize dusts known to be dangerous to health. In metal mines, dust must be removed or its formation prevented. The introduction into coal mines of suitably chosen stone dust to prevent explosions is commendable because of the immensely greater menace of explosions than of any probable health feature in rock dust, provided the latter is carefully selected.

The dust menace in metal mines is at its worst:

1. To workers in comparatively dry places in siliceous ore when dry drilling upper holes in unventilated places, especially if the air is above 80° F. (27° C.) and 85 per cent. relative humidity. Dry drilling of horizontal or downward holes by drills of the jackhamer type is little, if any, less dangerous than dry drilling of the upper holes. Several types of water stopers have been perfected for drilling the upper (even vertical) holes, and these not only prevent dust formation but are distinctly superior to dry drills in speed of drilling. Adequate currents of moving air at points of maximum dust formation either remove the dust or dilute the dust content of air below the danger point.

2. To workers in comparatively cool dry places where air is not kept moving by ventilation, especially where workers are on a contract basis, hence working at top speed. The fine particles of dust remain suspended in still air many hours and the rapid deep breathing of dust-laden air introduces a maximum number of dust particles into the respiratory passages of workers. The remedy is, in general, the establishment of adequate air circulation.

3. To workers in dry mines, or places, where blasting is done while men are on shift; the concussion due to blasting throws into the air clouds of fine dust, which remains suspended for several hours unless the air circulation at the working places is adequate.

The principal measures for the prevention or removal of dust in metal mines are as follows:

1. Eliminate dry drilling, using water-feed drills with hollow steel, there now being successful water drills for all purposes including the drilling of upper holes. In general, the system of mining should be devised so as to restrict to a minimum the drilling of upper holes.

2. As far as possible, blasting when men are on shift should be avoided. If the blasting is absolutely necessary, in dry siliceous material, adequate air currents to remove both the fumes of the explosives and the fine dust floating in the air should be provided. Timbers, walls, and floor adjacent to the point of blasting, in a dry place, should be thoroughly sprinkled before and immediately after blasting; the muck pile should be well sprinkled at all times.

3. Dusty places, such as manways, chutes, loading or dumping places, downcast haulage shafts, etc. should be kept damp by permanent

water sprays. Dusty haulage roads, shaft or other stations, etc. should be kept thoroughly sprinkled in some manner, such as by hose and nozzle as when sprinkling a city lawn, but the device must wash down the walls, timber, etc. as well as wet the floor.

4. The introduction of large quantities of water by the use of wet drills, sprinkling, etc. will cause the underground air to be practically saturated with moisture, a condition generally held to be harmful. However, high humidity is of little harm, even in still air, until the temperature is above 80° F. (27° C.), and if the air has a movement of a few hundred feet per minute, practically saturated air is readily endurable underground with temperatures up to or even over 90° F. If the water used for drilling, sprinkling, etc. underground is pure water piped from the surface (as it should be), the use of the water in hot mines will, in general, have a tendency to reduce the high air temperatures. Mining companies cannot, in general, expect to get full coöperation of miners in the use of wet drills in raises and stopes unless the water is piped to the machine. The men are reluctant to carry water in cans, barrels, etc. up ladderways, or even to haul it up by rope and pulley.

5. Where formation of dust cannot be prevented, one of the most efficient methods of removal is to sweep the dust-producing places with 1000 cu. ft., or more, of air per min., thus removing the dust-laden air from the places where men work and transferring the dust (simultaneously vastly diluting the proportion of dust particles in the air) to parts of the mine not frequented by workers, and ultimately removing the dust from the mine or, if the walls and floor are damp, depositing it along the walls of the air passages.

Tentative Conclusions and Recommendations as to Metal-mine Ventilation

Metal mines, in general, make comparatively little effort to provide adequate ventilation; namely, to supply at working faces and places currents of pure air circulating at sufficient velocity to allow employees to work at maximum speed without injury to health Many metal mines have flowing through the main air courses much more air than is necessary to provide the best ventilation, yet do not adequately ventilate more than 25 per cent. of the actual working faces or places.

Most of the older metal mining men consider ventilation a useless fad or an impossibility, and little progress can, in general, be effected as long as these men remain in charge.

Mines relying on natural ventilation are likely to be inefficiently ventilated during a great portion of the year, or when underground rock temperatures and surface air temperatures are about equal. In case of underground fires, the mine relying on natural ventilation is unable to control the air currents.

Financial Value of Good Ventilation

In addition to providing safer and more healthful working conditions for underground men, the establishment of good ventilation pays well in dollars and cents.

1. One cool mine, relying on natural ventilation, had good air circulation in midwinter but practically none in spring, summer, and fall, during which months there was not sufficient oxygen in the mine air to support combustion of a candle flame. While adjoining fairly well ventilated mines had an oversupply of men, this mine was always short of labor. After several years, a fan with about 30,000 cu. ft. (849 cu. m.) per min. capacity was installed; immediately this mine became a favorite with the men and its output soon jumped from a fraction over 2 tons to over 7 tons per man employed per shift.

2. In another mine relying on natural ventilation, the air was so hot and so smoky that men were continually being gassed (some fatally), and compressed air was used to ventilate. After the installation of fans with systematic coursing of air to the working places, compressed-air consumption was cut in half (the saving in air compression paid for the entire cost of the ventilating system) and the output per man per day nearly doubled, gassing was unknown and the accident rate, in general, fell perceptibly.

3. When a mine with one shaft and no air circulation and most workings below the 2000 ft. (609 m.) level with hot, humid, stagnant air was purchased, the new owner practically suspended ordinary mining operations and concentrated nearly all the working force on the construction of another shaft with suitable fan and underground air courses; the cost of the new work (solely for ventilation purposes) was well over $50,000, but the operating saving quickly returned the entire amount invested.

Air in metal mines (according to data from several hundred samples collected under various conditions in various mines in many states) is, in general, as free from chemical impurities, dangerous or unhealthful gases, etc. as in the much better ventilated mines of the coal regions, especially in the main return or discharge currents, yet at the face of blind-end drifts, crosscuts, raises, winzes, and stopes the air is depleted of oxygen, or charged with such quantities of CO or CO_2, or both, as to cause headache, nausea, etc. and hence greatly decrease the working efficiency and ultimately undermine the health of the men.

Elements Entering into Metal-mine Ventilation

The main elements entering into metal-mine ventilation, as viewed by the writer are, in the order of importance, air movement or velocity, air quality or chemical composition, air temperature, and air humidity.

1. In by far the greater number of underground places found to have

poor ventilation, the main underlying reason was lack of movement. It was found that if the velocity was adequate (100 ft. per min., or slightly over, in cool places and 400 or 500 ft., or over, in hot places, 70° F. or above), there was comparative comfort when temperature and humidity were high, even if there were slight amounts of chemical impurities in the air. It is the opinion of the writer that at least 90 per cent. of the inefficiency of present-day metal-mine ventilation is due to lack of movement of air at places where men work. Men have been seen to go from a stope with stagnant air to a level to "cool off," yet the moving air in the level was several degrees hotter and much more saturated than that of the stope.

2. While, in general, the quality of metal-mine air is good, there is frequently found at the working faces air containing over 0.25 per cent. CO_2 or as high as 0.10 per cent. CO, especially over muck piles, or below 17 per cent. in oxygen. If any one, or all, of these conditions exist in stagnant air, the workers are likely to suffer from headache or nausea and hence work with decreased efficiency and refuse to work full time. This is especially probable if the temperature is above 70° F. (26° C.) and the humidity above 85 per cent. One or more of these conditions is usually present at dead-end faces, unless especial means are taken to renew the air.

3. High temperatures in places containing stagnant air are most harmful to health and working efficiency, especially if the humidity is over 85 per cent., and particularly if much fine dry siliceous dust is present. However, if the air has a velocity of a few hundred feet (or more) per minute, temperatures well above 90° F. (32° C.) are readily endurable and probably are not unhealthful, even with high humidity; and pure circulating air quickly dilutes the dust content of air at a working face to safe limits.

4. High humidity of underground air is not disadvantageous to health or working efficiency until the temperature is over 70° F. (26° C.), and not then if the air has a velocity of several hundred feet per minute. It is the writer's opinion that with this air movement practically saturated air well above 90° F. will allow a man to do hard work without discomfort or ill effect.

Factors Affecting Temperature, Humidity, and Purity of Mine Air

Mine-air temperature is affected most directly by rock or underground-water temperature, and by oxidation of ore and timber, with other minor causes, as follows: Outside-air temperature, mine fires, friction of air in air passages, firing of shots, movement of ground, heat from lights and breathing of men and animals, heat from machinery, etc.

Air underground (especially if still) will, in general, take essentially the temperature of the surrounding rock, though dripping water will more quickly give its temperature to surrounding air than will the

surrounding rock. However, the passing of considerable quantities of moving air will tend ultimately to change the rock temperature to the temperature of the moving air. Where much timber is used in a region of stagnant air or where air currents are sluggish, especially if the timber is crushed and decaying, there will be much heat given off due to chemical action; this is also true where there is present much finely divided sulfide ore, especially copper sulfides, in a crushed zone. A combination of sluggish air currents with crushed timber and sulfide ore is apt to result in spontaneous fires.

High relative humidity (over 85 per cent.) is generally found in metal mines after leaving the main intake air courses. At least 90 per cent. of the working faces were found to have a relative humidity above 85 per cent., irrespective of the climate of the region in which the mine was situated or of the time of day or season of the year.

The main sources of vitiation of underground air are: Oxidation of ore or of timber giving off CO_2 and taking up oxygen; issuing of gases such as CO_2, nitrogen, etc., from adjacent strata; fumes from explosives giving off locally dangerous amounts of deadly CO, or oxides of nitrogen as well as considerable quantities of CO_2 and possibly other gases; breathing of men and animals; oxygen consumption by miners' lights; possible impurities in the intake air from smelter fumes, boiler-house gases, etc.; and possible impurities in compressed air, which contributes from 10 per cent., in well ventilated mines, to 30 per cent. of the entire air supply, in poorly ventilated mines. Though the composition of the main-return air of most metal mines is as good as that of coal mines, at the face of many blind ends the quality of the air is poor and at times dangerous.

Disadvantages of Using Compressed-air Blowers

In general, blind ends of crosscuts, drifts, raises, etc. are ventilated only by compressed-air blowers. The ordinary compressed-air blower (the open end of an air hose) delivers about 100 cu. ft. (2.8 cu. m.) of air per min., which will supply the ordinary drill but will not cool the air at the working place nor remove the bad air; workers at muck piles depending on compressed-air blowers frequently are nauseated. The temperature of the compressed air issuing from the end of the hose is little, if any, lower than that of the surrounding air, and as the blower works four or five times as long as the drill, it uses four or five times as much compressed air as the drill. In hot mines, this excessive consumption of compressed air reduces the air pressure to below 70 lb. (frequently below 50 lb.), and drilling is made ineffective and frequently must be suspended. Moreover, it costs about 100 times as much to compress a given quantity of compressed air (say 1000 cu. ft.) as it would to supply that amount by ordinary fan ventilation.

Common Easily Corrected Faults

Failure effectively and promptly to seal worked-out abandoned places results in loss of much pure air and air velocity to active workings; allows the lost air to enter the abandoned workings, where oxygen is absorbed by timber and CO_2 generated, and temperature and humidity are increased, to be taken later into active workings, to the detriment of employees. Besides, the absorption of oxygen with sluggish air flow in abandoned regions is likely to cause spontaneous fires. Sealing such abandoned places (cheaply and effectively done by guniting) will conserve the air supply for use in active workings in unvitiated form, and will tend to prevent spontaneous fires.

Currents of comparatively cool, pure air frequently pass within a few feet of places where men work about 10 or 15 min. in hot, stagnant, humid air, and then spend 20 to 40 min. "cooling off" in the nearby air current, which could readily be diverted to the working face.

Mines having hot rock but cool water, either from the workings or piped from the surface, allow the air to assume the temperature of the surrounding rock, when sprays of the cool water would give cool air to workers.

Frequently small fans with canvas or galvanized-iron pipe are provided to force air to the working faces, but the end of the pipe is left 100 ft. or more from the face, and the worker at the face is left to suffer from hot, stagnant air, and frequently headache and nausea from fumes from the muck pile.

Air courses are frequently piled with lumber, timbers, etc. practically cutting off air flow. In main shafts, cages or skips are left standing in such manner as practically to plug the shaft. In main haulage drifts, also acting as air courses, trips of cars are left standing so that air flow is practically nil. Doors are frequently left open for hours at a time, depriving entire sections of the mine of air circulation; only too frequently doors are of such faulty construction, or in such poor repair, that more air leaks through than is held in check.

Connecting Ventilation Systems of Different Mines

The general metal-mine practice (and frequent coal-mine practice) is to ventilate one mine through another. While this is cheap and convenient, it is inefficient and dangerous, as the mine receiving air from another usually has hot, humid, vitiated air frequently impregnated with gases, fumes from explosives, etc. and with dangerous dust. There have been many deaths in metal mines, as well as coal mines, from fumes from a fire in an adjoining mine (due to reversal of air currents) even when under ordinary ventilation conditions the mine with the fire is on the return air.

Relation of Ventilation to Fire Protection and Prevention

One of the most important considerations in connection with metal-mine ventilation is its relation to fire protection and prevention. In addition to the advantage of having fans with provision for readily reversing air currents, fire protection is greatly advanced by a systematic splitting of air currents, and by having a system of well constructed doors in openings along main intake shaft or air course, such that the main intake shaft or air course may be positively isolated from the remainder of the mine in case of fire in the intake. These doors (and in fact all doors underground) should be held shut by a latch, system of weights and pulleys, etc. A door held shut only by the air pressure is dangerous, as air currents are likely to be reversed, either naturally or mechanically, at time of fire. Automatic doors should have devices that will close the doors and hold them shut when they are released after being opened.

Data Obtained by Ventilating Experiments

Valuable data have been obtained during the past few years by progressive metal-mining companies experimenting in ventilation problems and with various kinds of equipment.

1. Small fan and canvas pipe units have been developed for delivery of 1000 to 5000 cu. ft. (28 to 141 cu. m.) of air per min. to workers at blind-end faces of drifts, crosscuts, stopes, raises and winzes, greatly improving working conditions and decreasing costs, labor turnover, etc. To avoid the heavy expense due to rapid decay of the canvas tubing, recent experimental work has developed a process by which the cloth is treated, and decay materially reduced or at least delayed.

2. Partly for fireproofing, partly to reduce friction of air in shafts and other main air courses, there have been developed several methods of smooth-lining by use of the cement gun, with the result that the air flow is practically doubled with the same amount of power.

3. Similarly, it is found that smooth-lined circular shafts will deliver over three times as much air as the same cross-sectional area in a rectangular, exposed-timber shaft as usually constructed.

4. Use of small fans forcing air at high velocities through galvanized iron or canvas tubing causes rapid increase in temperature, together with simultaneous decrease of relative humidity. While the increase of temperature is a disadvantage, the decreased humidity and rapid movement much more than offset any disadvantages.

5. Use of cool water sprays is effective in reducing temperature of underground air, and when carried on in conjunction with fan units giving adequate velocities, workers in hot, deep mines are given decided relief.

Recommendations as to Metal-mine Ventilation

All metal mines should be ventilated mechanically, using a fan in a fireproof housing, placed preferably on the surface, and having a capacity of 50,000 cu. ft. or more per min. for a fairly large mine, or one with high rock or water temperatures, or having gas flow from strata. The fan should operate 24 hr. per day and should be able to deliver against a water-gage pressure as high as 5 in. There should be provision, by suitably arranged doors or otherwise, for the prompt reversal of air currents desirable.

The mine should be divided into absolutely separate air splits, so that the intake air for a split will have a velocity of 200 to 500 ft. (60 to 152 m.) per min. in the main air courses, circulate through a limited number of air places, and be removed to the surface before the temperature, humidity, dust, and gas content become so high as to be harmful to the health or efficiency of the workers. The establishment of absolutely separate splits will also, in case of mine fire, confine dangerous fumes to one split, and prevent the spread of these gases through the other mine workings or into other mines.

A special effort should be made to conduct moving air to the working places. The mining system (especially future work in a well prospected mine) should therefore be planned to give an intake air course and a return air course, as in coal-mining practice, by driving parallel workings in pairs with frequent connections and closing all the connections except the one nearest the working face. Where necessary to drive single drifts, crosscuts, raises, winzes, etc., provision should be made for adequate mechanical ventilation, preferably with a small fan forcing air through canvas or galvanized-iron pipe. Canvas pipe should be preferred as it is more flexible and more likely to deliver the maximum fan capacity at the working face; by using a "blasting length," the air current can be kept close to the face at ordinary times.

While suction ventilation of blind ends will more quickly remove explosive fumes from the working face, at ordinary working periods it does not give the cooling and exhilarating effect of the air as it issues from the end of the tube in the pressure system. Moreover, by delivering 1000 cu. ft., or more, of air per min., a fan and canvas pipe system will clear an ordinary working place of fumes (or at least safely dilute it) within 10 or 15 min. The compressed-air blower delivers only about 100 cu. ft. of air per min. and is dangerous to rely on for the removal of fumes.

Doors should be so placed as to control air currents, prevent leakage, and isolate the intake or the return shaft. All doors supposed to be kept closed should be held closed positively by some definite method. The placing of well made doors is a vital part of safe and efficient ventilation practice.

As workings are abandoned, they should be effectively sealed to prevent spontaneous fires and to conserve the air supply for active workings. Gunite over boards and metal lath or chicken-wire or over gunnysacking and chicken-wire gives a substantial and tight seal.

Where possible, each mine should be ventilated absolutely independently of every other mine with both return and intake air courses (shafts, drifts, etc.) to the deepest or farthest workings. If possible, main intake and return air courses should not be used for other purposes, such as haulage.

Air courses should be kept free of obstructions. Cages and skips should be held above the shaft collar when not in use. Levels used as air courses should be kept free of piled timber, standing cars, locomotives, etc. At intervals, raises should be provided between levels or into stopes, with platforms and other obstructions removed to allow free circulation of air. Where shafts, raises, etc. are of small cross-sectional area, they should be smooth-lined, as this practically doubles the possible air flow for the same pressure.

The coal-mining practice of using deflecting curtains of fireproof brattice cloth, or of using lined brattices, overcasts and deflecting doors in raises, might be adopted to advantage. Where cool water is available, it might be used to cool air currents by sprays in hot drifts or to cool the air and increase its flow in downcast shafts, raises, etc. With the efficient use of cool-water sprays and a high air velocity, there is no reason why men may not work under comfortable and healthful conditions until rock temperatures approach 135° F. (57° C.), which is much higher than temperatures in the deepest workings of any mining region of the United States, with the possible exception of one camp that is not heavily worked at present.

By far the most essential feature in providing adequate and efficient ventilation of metal mines is constant, intelligent supervision, which can be had only by assigning all ventilation work to a wide-awake progressive engineer.

Conclusion

The above data, conclusions, and recommendations embody essentially the personal experience and views of the writer and are offered with the idea that later experience and data may cause some changes; however, the data and conclusions are given after over 20 years' experience and observation in coal and metal mines in over twenty-five states of the Union.

DISCUSSION

H. M. Landis, Philadelphia, Pa.—Until comparatively recently, all dusts were grouped under the same heading. Of course they were subdivided into organic dust, mixed dust, and inorganic dust, but there

was little or no distinction as to their relative harmfulness. This distinction has been established in the last few years, and has a certain relation now to preventative medicine and industrial problems. There is no pathological proof as to the harmfulness of organic dust, which never produces any injuries of the lungs. Inorganic dust, on the other hand, if inhaled under the proper conditions and for a sufficient length of time, will almost certainly produce perfectly definite pathological changes in the lungs, so definite that you can predict absolutely what is going to happen.

A potter for example, can be in the trade for about 15 years before he begins to show any evidence of injury to his health. More often it is about 20 years, and it will be about 30 or 35 years before there is definite, serious change. Even then, the ordinary examination in the early stages will show nothing by physical signs, but the X-ray will show, as in all cases of inorganic dust, a most serious distorting disease of the lungs.

Silica produces a more acute inflammatory condition of the lungs than any other of the inorganic dusts. The idea has been advanced that silica gives rise to some chemical reaction. This hypothesis has never appealed to me, because, so far as I know, the only chemical substance that has any effect on silica is hydrofluoric acid.

It hardly seems possible to me that there is any chemical fluid in the body, which can be involved, that would have a corroding influence, or would be apt to set up a chemical reaction if brought in contact with materials of this sort. Furthermore, the South African observers were able to show, from histological studies of the lungs, that these fine pieces of silica, the sharp points, which would be the ones first influenced by such chemical action, were unchanged.

The changes that take place in the lung as the result of inhaling inorganic dust are definitely progressive. In some men they occur early and in others they are more or less deferred; every now and then one is encountered who seems to be resistant to the effects of the dust.

So many men in the granite cutting industry have died or become incapacitated as the result of intense exposure to inorganic dust that a serious attempt is being made to introduce preventive measures in order to save the industry. When the investigation of the granite cutters was started, it was thought that the industry was productive of an undue amount of tuberculosis, but it has been found that many of these cases are pure pneumoconiosis. The symptoms are almost identical with those of tuberculosis and even an experienced X-ray man may have difficulty in telling whether or not tuberculosis is also present.

Doctor Lanza has pointed out that the living conditions of miners show that infection occurs in many of the men, not as a result of the

industry, but of their living habits, and of their houses becoming infected. I know of no condition that demands intensive study on the part of large corporations and engineers more than the protecting of the men against inorganic dust. The real work that must be done today is in educating the workmen; they are prone to become careless about utilizing the appliances at hand. Also, they are apt to be rebellious, particularly in the matter of physical examination, assuming that somebody is trying to find something the matter with them, so that they can be put out of work.

A. J. LANZA.—In the case of hard-rock dust, the danger consists in the amount of silica in the dust, and the extent of exposure. It is purely a mechanical proposition. A man may work three or four months a year in a silicious dust without injury while a man working twelve months a year in a moderately silicious dust will be a victim. Exposure to silicious dust for any great length of time is inevitably harmful.

It is unfortunate that the prevention of dust—and therein lies the prevention of disease; once dust is in the air it is almost impossible to do anything with it—is not as easy in practice as it seems on paper. It would seem, theoretically, that the liberal use of water would be simple, but in mines situated at a high altitude, during the winter, the liberal use of water is apt to cause freezing of dirt when hauled to the surface; besides, the average miner would rather breathe rock dust, and know he was going to die of consumption, than to get wet and fear that he was going to get rheumatism.

The use of respirators for protection against dust is fallacious. The average size of dust particles that penetrate into the lungs is about five microns in diameter. Larger particles remain in the upper respiratory tract. A respirator of a sufficiently fine mesh to filter out of the air particles of from two or three microns to ten microns in diameter, would offer so much resistance to breathing that a man could not work with it.

It is in the application of water that our safety lies. That idea has been carried to a high degree of perfection in South Africa. There blasting is never done in hard rock mines without using a water blast. This is an intensive water spray that is turned on just before the shots are fired, so that as the shots go off the whole thing is buried under an immense spray of water.

GEORGE E. COLLINS, Denver, Colo. (written discussion).—My experience is that in mines opened by adit level with connection to one or more surface shafts—such as are common in Colorado—natural ventilation is usually effective; the natural current is continuously at work, whereas in mines where work is carried on for only one or two shifts artificial ventilation is commonly suspended when the mine is idle.

The greatest single improvement in connection with dust is to confine work to a single shift, so that there is ample opportunity for fine dust to settle in the interval between shifts. The other advantages to the health and comfort of the men, arising from day-shift work, are such that I have always planned, whenever possible, to concentrate underground work on day shift.

My observation has been that the use of wet machine drills does not protect men from contracting miners' consumption. The striking differences between different mines and different districts in respect to the relative number of men contracting this disease suggest that much remains to be learned as to its causes.

I do not agree with Mr. Harrington's recommendation to use an underground ventilation fan with tubing solely as a blower. It should be connected so as to blow or exhaust at will: the former for use when men are at work, the latter to remove dust and gases after blasting. Canvas pipe is convenient to handle, but it cannot be used for exhausting. As a rule, galvanized-iron pipe is preferable, with a couple of lengths of canvas pipe for use at the breast. The direct use of compressed air for blowing back gases after blasting is ineffective, as it often drives it merely a few hundred feet from the face. But the compressed air, used as a jet in a galvanized-iron pipe to induce an exhaust, is an effective, although rather expensive, aid to ventilation.

All ventilation work should not be taken completely out of the jurisdiction of lesser mine officials. The essential point is to educate foremen and shift bosses to realize the value of good ventilation. I can remember when horizontal brattices in drifts (to induce ventilation) were far more general than now. The constant tendency to relieve the men and their immediate leaders of all duties and responsibilities removes the necessity and incentive for intelligent work.

Alaskan Coal Fields

By George Watkin Evans, Seattle, Wash.

(New York Meeting, February, 1921)

During the past ten or twelve years, the average reader of newspaper and magazine articles has been led to believe that enormous deposits of high-grade coal exist in the northland and that these can be mined for practically nothing. But much of the reputed glitter and attractiveness of Alaskan coals disappears when viewed at close range.

Many areas of coal-bearing rocks are widely distributed geographically from Cape Lisburne, on Bering Sea, to Admiralty Island, south of Juneau. Bituminous coal is found at Cape Lisburne, on the Bering Sea, at Five Finger Rapids on the upper Yukon, and at Herendeen Bay and Chignik Bay; lignite is found on the Kobuck River, along the Yukon River from points near the mouth to Tonakat, near Rampart, at Dawson, on the east shore of Cook Inlet, and in the Tanana Valley, 50 mi. south of Nenana. The Susitna and Matanuska Valleys contain coal ranging from lignite to anthracite, and in the Bering Lake district, there are coals ranging from semi-bituminous to anthracite.

No effort will be made to describe all of these fields, but the writer will confine himself to the four principal fields, namely, the Nenana field, in the interior of Alaska, tributary to the Fairbanks district; the Matanuska field, in the Matanuska Valley; the Kachemak Bay field on the east shore of Cook Inlet; and the Bering River field, east of the mouth of Copper River.

Nenana Coal Field

The Nenana field occupies an area east of the Nenana River 70 mi. (112 km.) southwest of Fairbanks and approximately 50 mi. from the town of Nenana, which is located at the junction of the Alaskan Railway and the Tanana River. It is on the northern flank of the Alaskan Range and occurs within the foothills of that district. The elevations range from 1000 to 4000 ft. (304 to 1219 m.). Future developments may prove the field to extend from a considerable distance west of the Nenana River to the headwaters of the Tanana River.

The coal measures are of Tertiary age and are probably Eocene. They are not badly deformed by folding, but portions of the field show

a small amount of faulting. Along Lignite Creek the measures are regular and should be favorable for mining. Eight or more coal beds are exposed in a cross-section at Lignite Creek. The lowest is over 40 ft. (12 m.) thick and two others are over 30 ft. thick. The coals are lignitic and, as mined, contain a high percentage of moisture. One analysis showed: moisture, 35.10 per cent.; volatile matter, 25.10 per cent.; fixed carbon, 34.60 per cent.; ash, 5.20 per cent.; calorific value 7000 to 7800 B.t.u. This coal in its natural state contains a high amount of moisture and is of low rank.

The continuity of the coal beds and the small amount of deformation makes the field attractive from a mining standpoint and, because of the great thicknesses of the coal beds, should permit of lower mining costs than any of the other Alaskan fields. It is not improbable that in portions of the Lignite Creek area the coal can be mined by scrapers or steam shovels.

Much has been said about the need of fuel in the interior of Alaska; in the Fairbanks district, prior to the war, wood sold for from $9 to $15 a cord and even at these prices it was difficult to secure suitable wood within certain districts. With a cheaper fuel available, there is reason to believe that many of the low-grade placer claims in the Tanana district, which are now idle, could be successfully operated.

The coal of the Nenana field might be utilized by partial carbonization and briqueting of the residue. In the briqueted form, the product would be of much higher grade than the raw coal and would bear transportation costs to better advantage. Another method would be to install a central power plant in the coal field itself and distribute steam-generated electricity to the placer mines of the north. This would be an admirable scheme so far as power is concerned, but would not meet the demand for fuel for heating purposes.

The government railroad has been built into this field and transportation is provided into the Tanana Valley; one small mine has been opened to supply the needs of the railroad and the town of Nenana.

Matanuska Coal Field

The Matanuska coal field lies between Moose Creek and Thaneta Pass. Disconnected coal areas are distributed from the head of Cook Inlet to Thaneta Pass, but Moose Creek is usually recognized as the western boundary. The field lies northeastward of Anchorage, with which it is connected by a branch from the main line at Matanuska Junction.

The elevations range from about 500 ft. (152 m.), in the southwestern portion, to 3000 and 4000 ft. in the northeasterly area. The hills and

mountains rise from the Matanuska Valley toward the Talkeetna Mountains on the north and the Chugach Mountains on the south. The Matanuska River drains the entire district. The principal tributary streams are Moose Creek, Kings River, and Chickaloon River.

The coal-bearing strata are Tertiary in age and are probably Eocene. The coal beds in the area west of Moose Creek are lignitic; to the eastward they become anthracitic in a small area east of Chickaloon.

The coal beds have been greatly deformed within both the Moose Creek and the Chickaloon district. In the Moose Creek-Eska Creek areas a great deal of faulting occurs which has greatly handicapped development. At Chickaloon, the beds are folded and are highly faulted; in addition they are intruded by igneous rocks.

Within the Eska Creek area, which lies east of Moose Creek, eight or more beds have been found, six of which have been developed to some extent. These beds range in thickness from 2 ft. 6 in. to 14 and 15 ft. (0.75 to 4.5 m.). The thicker beds contain numerous bands of impurities. In the Chickaloon area, six or eight coal beds have been discovered; they range from 3 to 16 ft. in thickness, but carry a rather high percentage of impurities. This coal will be of value mainly for local consumption. An analysis of the coal in the best bed is: moisture, 5.56 per cent.; volatile matter, 36.52 per cent.; fixed carbon, 51.32 per cent.; ash, 6.60 per cent.

Within the Chickaloon area a better grade of coal has been found; in fact, the coal mined by the Naval Expedition of 1913 was rated 98 per cent. as efficient as eastern Pocahontas coal for steam-generating purposes. An average analysis of a sample of 800 tons of the smaller sizes of coal from one bed showed 2 per cent. moisture, 20 per cent. volatile matter, 70 per cent. fixed carbon, and 8 per cent. ash. The larger sizes from this same bed contained a much higher percentage of ash. When properly cleaned, the coal from the Chickaloon area is an excellent steam coal and is suitable for coking and blacksmithing.

The mining conditions at Eska Creek are somewhat similar to those in western Washington, a region that is folded and faulted to as high a degree as any other commercial coal field in the United States. During 1917, the average production per man per day was 0.47 ton; in 1918, 1.5 tons; and in 1919, 1.65 tons. No corresponding records are available for the Chickaloon district but there is good reason to believe that this district will not prove to be any better, though the No. 8 bed at Chickaloon might show better averages. Both the Eska Creek and the Chickaloon districts will be expensive to develop and operate.

The greatest need for the development of the Matanuska and other Alaskan coal fields is to supply the demands of the area tributary to the field itself. In addition, the Alaska Railway will require a reasonable amount of coal. Should a smelter be constructed in the western part

of Alaska, close to tidewater, coke made from Chickaloon coal might be utilized for smelting purposes. If produced in sufficient quantities, Chickaloon coal might be marketed for blacksmithing and to the United States Navy.

Considerable doubt exists as to the total minable tonnage within the high-grade portion of the field at Chickaloon. The development work done within the past three years indicates only a limited tonnage. At the Eska mine, gangways have been driven on five coal beds in the area west of the creek and an equal number of gangways on beds east of the creek. In 1917, 13,869.7 tons; in 1918, 54,912.88 tons; and in 1919, 39,975.85 tons were mined. At Chickaloon, a slope about 600 ft. (182 m.) deep has been sunk on one of the beds and numerous rock tunnels have been driven.

Kachemak Bay Coal Field

The Kachemak Bay district is the southeasterly extremity of a large coal area that extends along the east shore of Cook Inlet. The elevations range from tidewater to about 1000 ft., on the plateau north of Kachemak Bay. In general, the district is a moderately sloping plateau overlooking Kachemak Bay and a part of Cook Inlet.

The coal beds are Tertiary and probably upper Eocene. The beds lie at gentle angles, for the greater part, dipping to the northeastward toward the plateau. There are a few faults within the field, but the structure is generally simple and the faulting not of large displacement.

Seven coal beds have been reported at Bluff Point, but these are probably not the complete series of coal beds in this district. Three of the beds are approximately 4 ft. (1.2 m.) thick, one is 6 ft. thick, and the other three are from 2 to 3 ft. thick. An average analysis for the coals is: moisture 21.54 per cent., volatile matter 39.1 per cent., fixed carbon 30.26 per cent., ash 9.1 per cent.

The ease with which the coal can be mined and its accessibility to water transportation should make this district one of the most attractive to the financier, yet it has received the least attention. It is true that the coal is of low rank, but a greater number of British thermal units per dollar can be mined and delivered to the consumer than in any other coal field in Alaska tributary to the Pacific Ocean. A mine producing about 100 tons per day is in operation at Bluff Point during the summer season. The coal is largely consumed at the canneries and on the boats used by the cannery companies.

Bering River Coal Field

The Bering River coal field is located from 50 to 80 mi. (80 to 128 km.) easterly from the town of Cordova. It occupies a portion of the foothills

of the Chugach Mountains east of the mouth of Copper River. This is the coal field that received so much publicity during the Ballinger-Pinchot controversy and in it are located the much-discussed Cunningham group of coal claims.

This coal field is approached more easily from Katalla than from Cordova and lies from 15 to 40 mi. northeasterly from the former place. Recently a truck road was constructed connecting a portion of the field with a shipping point near Bering Lake; a railroad has also been built from a point on Bering River to the northeasterly end of the coal field but it is not in operation at the present time. The elevations range from tidewater to between 5000 and 6000 ft. in the extreme northeasterly portion of the field; the hills are rather precipitous.

The coal beds within this field are either Upper Eocene or Lower Miocene. From a structural standpoint the field throughout is complex and the disturbances increase from west to east. Folding has been intense; in addition there are many faults and in the eastern end intrusive rocks add to the complexity of structure. A large tonnage of coal, however, apparently exists above water level.

It is impossible, from the present available information, to state exactly how many coal beds are within this field. Within a given cross-section, seven separate beds have been enumerated but probably these are not all. Because of the excessive faulting and folding, it is believed that beds have been repeated a number of times within the same cross-section. There are hundreds of outcrops in this field; in one tract alone, consisting of a little over 2000 acres, 250 separate coal outcrops have been observed. These, no doubt, are repetitions and represent elongated lenses of coal rather than distinct separate beds. Unquestionably a greater number of outcrops occur within this coal field than in any other coal field in Alaska so far developed, but this large number of outcrops indicates uncertainty of continuity of the beds.

The coal of this field is no doubt the highest grade so far discovered on the Pacific Coast. In the westerly portion, the coal is semi-bituminous, near the central area it becomes semi-anthracite, and in the extreme eastern end anthracite has been found. Recently the writer made exhaustive tests and found that coal with approximately the following analysis can be produced in a properly constructed and operated plant: moisture 1.0 per cent., volatile matter 15.0 per cent., fixed carbon 78.0 per cent., ash 6.0 per cent. Several of the "floats" of the samples taken during the past summer contained less than 2 per cent. ash. But the reader must not think that the beds contain coal of uniform character between the walls because this is not true—the beds as a whole contain a large amount of impurities associated with the coal. The samples collected were obtained in much the same manner as the coal will be mined in regular practice and exhaustive float-and-sink tests were made

on a large scale. It was proved that the coal is clean and of high grade and will make a good separation from the impurities that will accompany it as mined. Although the coal is uniformly clean from one end of the field to the other, the associated impurities within will require care in mining and attention to the proper construction and operation of the coal-cleaning plant.

In 1912, between 800 and 1000 tons of coal were mined from the Cunningham property and tested on a United States battleship. The conclusion reached was that the coal is not suitable for naval use. It is probable that large percentages of shale were included in this test sample, as the coal and associated impurities resemble each other so closely that it is difficult to distinguish between the coal and the shale. A properly constructed and operated coal-cleaning plant would make possible good separation and there is little doubt that properly prepared coal from this field could meet the standards of the Navy test.

The coal in the western portion of the field is badly crushed and will not stand much handling without crumbling into dust. Within parts of this area, the coal will coke and it is claimed that coke of excellent quality has been made from coal from the Cunningham property. The writer is not thoroughly convinced that the coal of this field will make uniformly good coke. Tests made recently indicate that the coal coked very well, whereas in other tests it did not coke at all. Statements have been made that this field contains a large tonnage of high-grade coking coal, but the writer doubts the uniform coking character of the coal from this field as a whole. It is known that east of Clear Creek, from a point near the central portion of the field to the extreme eastern end, the coal becomes anthracitic and has lost any coking property that it might have had.

The cost of mining coal in this field will be high, and it is questionable if a large part of the field itself can be operated at a profit. The disconnected character of the coal lenses and the crushed character of the coal, as well as the surrounding walls, make the problem of mining difficult and hazardous. It would be necessary to do considerable rock tunnel work in proportion to work on coal. The stories told about the mountains of high-grade coal in this field that can be mined at little or no expense are absolutely false. The cost of mining coal in this district will be as great, if not greater, than in any of the coal fields now operating in the western part of Washington.

With fuel oil disappearing from the fuel markets of the Pacific Coast, especially the north Pacific Coast, indications are that coal from some field must be used. The coal within the Bering River field is high grade and would make an excellent steam fuel, for which reason this coal should stand a good chance of supplying these needs. At present, there is a market for 30,000 or 40,000 tons of coal a year along the

Alaskan coast, but if fuel oil is replaced by coal, there will be a market for over 150,000 tons a year.

At present two companies are doing development work in this field. One is operating on a lease in the area between Lake Kushtaka and Lake Charlotte and the other is operating in the Carbon Mountain district in the area east of Canyon Creek. The latter property is connected with Bering River by a railroad but no appreciable amount of coal has been shipped. The property that is being developed between Lake Kushtaka and Lake Charlotte is connected with a point on Shepherd Creek by a truck road. This company has not yet reached a producing stage, but it is expected that upwards of 50 tons per day will be produced during the summer of 1921.

Market for Alaskan Coal

When developed, Nenana field will supply the needs of Fairbanks and the surrounding country. At present the market for coal in the Fairbanks district is rather limited, perhaps not more than 30,000 tons a year is required; but if the placer mines of the Tanana and Yukon basin develop as rapidly as some people think they will, a much larger market for the coal from this field will be developed. In addition, the operation of the government railroad between Broad Pass and Fairbanks will require fuel.

The Matanuska field contains two grades of coal. The Eska Creek-Moose Creek coal is not of good enough quality to send to distant points. The Chickaloon coal, however, is of high grade and, if produced in large enough quantities at a reasonable price, should find a ready market for naval and blacksmithing use and for coking. The Alaska Railway will no doubt use this coal in preference to the Eska coal if it can be produced at a reasonable price.

The Kachemak Bay area, because it contains coal that can be mined at a moderate price and is also provided with water transportation, should be a factor in supplying the fuel need of the Cook Inlet country and the surrounding areas.

As to the markets for the coals of the Bering River and Matanuska fields, much will depend on the cheapness with which they can be mined and how they will stand transportation. The householders have been burning lump coal from Vancouver Island and it will be difficult to educate them to use the smaller sizes of high-grade coal from these fields. For this reason neither of these coals will find favor at first as domestic fuels.

For export trade it is doubtful if either coal will enter the markets of British Columbia or Washington except for special uses. At present no coal mined on Vancouver Island or in Washington is suitable for naval

use, and if the Bering River or Matanuska field produces coal sufficiently cheap and in large enough quantities, these coals will no doubt be used by the Navy in the future.

It is not at all likely that Alaskan coal from the Matanuska, the Kachemak Bay, or the Bering River field will be shipped into Washington, Oregon, or California for general steam or domestic use. Neither the Bering River nor the Matanuska field will produce a high-grade coal in lump form, and inasmuch as the people of the Pacific Coast are accustomed to using lump coal, it will be difficult to introduce the smaller sizes produced in the Alaskan fields.

Total Coal Areas in the Alaskan Coal Fields

It is estimated, by Stephen Capps of the United States Geological Survey, who mapped the Nenana coal field, that 165 sq. mi. (64 sq. km.) of land are underlain with coal-bearing strata in that district. This is probably a conservative estimate for the coal area of this district. No doubt, on further development, additional areas will be found. It has been stated that the beds of this field are thick, uniform, and regular, so there is no doubt that there is a large total tonnage of minable coal within this field.

George C. Martin, of the United States Geological Survey, estimates that there are 54 sq. mi. (21 sq. km.) of supposed coal-bearing formations in the Matanuska field proper and that probably an additional 24 sq. mi. might be regarded as extensions of supposed coal-bearing areas. The area between Eska Creek and Moose Creek undoubtedly contains a large tonnage of minable coal, but the area between Chickaloon and Kings River is somewhat doubtful; at least the more recent work done in this part of the field indicates considerable doubt as to the continuity of the beds between these two places. Work is being done at the present time by the United States Navy to demonstrate the amount of minable coal in the area lying between Chickaloon and Kings River. An additional area of coal-bearing strata lies eastward of Chickaloon and it is probable that, with further development, additional areas of high-grade coal will be found within this district.

The Kachemak Bay field contains a large total area of coal-bearing strata. The coal-bearing formations appear to extend to the eastward from Bluff Point to the head of Kachemak Bay and to the northward along the northeast shore of Cook Inlet for a considerable distance. It is only by extensive future exploratory work that the full extent of this coal field can be determined. The continuity of the coal beds within this area and the ease with which some of them can be mined indicate clearly that there is a large total tonnage of minable coal within this district.

The Bering River field contains approximately 22 sq. mi. of coal-bearing formation in the bituminous portion of the field and perhaps 25 or 30 sq. mi. of coal-bearing formation in the anthracite part of the field. Of the 22 sq. mi. or more of coal-bearing strata in the bituminous area, only a comparatively small portion will prove to contain coal that can be mined at a reasonable cost, for which reason the total tonnage of minable coal within this portion of the field will be much less than is generally supposed. This statement also applies to the anthracite part of the field.

Variability of Alaskan Coals

In the Nenana field the beds are normal, not much disturbed, and lignitic in character. In the Matanuska-Susitna field, especially in the vicinity of the Little Susitna River, the beds are normal and of lignitic nature. Eastward, the beds become more disturbed; the percentages of moisture in the coal decrease and the percentages of fixed carbon increase in the approximate ratio that the folding and faulting occurs. As the higher ranks of coal occur in areas of disturbance, they will be more expensive to mine. Within the Kachemak Bay field, the beds are lignitic or sub-bituminous but are regular in structure. In the Bering River field, the beds are compressed by folding and faulting to perhaps a greater degree than portions of the Matanuska fields; also the moisture content is less and the fixed-carbon content is greater than in the Matanuska. The greater compression in the eastern parts of this field over that of the western part has proportionately increased the fixed carbon content of the coal.

Coal Fields of the Pacific Coast

In considering the Alaska coal problem one must take into account the other coal fields tributary to the Pacific Coast. Washington produces between 3,000,000 and 4,000,000 tons of coal a year, and the coast district of British Columbia produced in 1919, 1,850,142 tons. In addition, considerably over 500,000 tons of Utah and Wyoming coal were used. The production in California is small and has little bearing on the situation. Vancouver Island and portions of the coast district of British Columbia, Washington, and Oregon have large areas of undeveloped coal lands; Wyoming and Utah possess large coal reserves and can continue to produce at the present rate for many years.

Until the present, California has been supplying large quantities of fuel oil for the steam markets of the Pacific Coast. The large companies are not making contracts to supply fuel oil for any considerable period of time, and, furthermore, the price of oil has advanced materially within the past two years. It is expected that within a few years Cali-

fornia fuel oil will play an unimportant part in the fuel markets of the Pacific Coast. Not only has fuel oil been supplying the fuel needs of California, Oregon, and Washington, but also the Province of British Columbia, and to a large extent, Alaska. The disappearance of California oil from the Alaska markets makes it all the more necessary to develop Alaska coal at this time.

Conclusion

Exploitation of the Alaskan coal fields began about 1903 and development continued, in a small way, in the Bering River field until 1908, when President Roosevelt withdrew the Alaska coal lands from entry. Some development work had been done in the Matanuska coal fields and a small amount of work had been carried on in the Kachemak Bay field. When these coal lands were withdrawn from entry, development work ceased. Since 1915, it has been possible to lease coal land in both the Matanuska and the Bering River fields; the Kachemak Bay and the Nenana coal fields have more recently been subdivided into leasing units.

Three attempts were made by private lessees within the Matanuska field to develop commercially valuable coal mines, but all three failed. The government has taken over one of the leases to supply its needs in railroad construction. Figures are not available covering the expenditures incurred in the development of the Matanuska field, but no money has been made in the development of these coal areas, when total expenditures are taken into consideration.

Several attempts have been made in the Bering River field to develop profitable coal mines, so that it appears that one of the companies in this field, and perhaps two, will develop coal mines of small production which will probably pay to operate. Taking the investment in this field as a whole, the coal mining venture has been a loss financially, and the individuals or organizations who have been putting their money into the field have been taking decided risks.

Compare the foregoing statements of fact with the information given out by ultra-conservationists during the Alaska coal controversy, to the effect that each and every coal claim in the Bering River field was worth in the neighborhood of $150,000. Instead of retarding private development in any of the Alaska coal fields, the government should have encouraged legitimate development in every way possible; it has assumed the latter attitude during the past few years. The writer believes that any man, or association of men, who has the courage to invest money in the development of Alaskan coal lands deserves every encouragement possible, and is entitled to all the money that can be legitimately made, for the risk is unusually great.

DISCUSSION

P. N. Moore, St. Louis, Mo.—Two years ago while waiting for a steamer, I was shown over the Chickaloon operations of the Alaskan Railway Commission by the engineer-manager, Sumner Smith. My observations of the coal confirm the conclusions of Mr. Evans. There is a small field of intensely tortured, broken coal, frequently cut by dikes. After my return I followed the reports of the Alaskan Railway Commission concerning this operation. The Chickaloon mine usually employed 30 to 40 men, and occasionally had an output of 125 tons per week. It is but fair to state, however, that a large part of the work was prospecting. This end of the field shows remarkable foldings. In one case a clearly visible outcrop, a 4-ft. coal seam, is doubled twice on itself; so that it is called a 12-ft. seam by careless observers. An official who visited the region last summer reported that they had a 12-ft. seam in that field. This portion of the field has been altered until the volatile matter is much reduced, and the coal has a good heating power. Whether it will coke or not, I cannot say. At present, further development of it is being conducted by the Navy Department under a special appropriation of $1,000,000 by Congress for the purpose of thoroughly testing the field. Whether the supply from it will ever reach any large amount is doubtful; but undoubtedly it will suffice for any local demand likely to exist in that portion of Alaska.

H. H. Stoek,* Urbana, Ill.—William Griffith, of Scranton, Pa., 15 or 20 years ago, published an article in *Mines and Minerals*, in which he called attention to the fact brought out in the paper, that the surface indications are misleading, and that the amount of coal to be expected is much less than prophesied through the newspapers at that time.

Geo. S. Rice,† Washington, D. C.—While I have not visited Alaska, some of the activities of the work there have passed under my general supervision.

When the Alaskan Engineering Commission needed coal, it tried to get it at Chickaloon, where previous expeditions had found that coal more nearly like that required by the Navy was to be found, but the Commission discovered that farther west, at Eska Creek, where the metamorphism had not been so great, that the coal lay rather more regularly in beds ranging from 3 to 5 ft. thick. These were opened and furnished the main supply for the Alaskan Engineering Commission. The maximum output was about 4000 tons in a month, which was more than the Alaskan Engineering Commission needed. So 30,000 or

* Professor of Mining Engineering, University of Illinois.

† Chief Mining Engineer, U. S. Bureau of Mines.

40,000 tons were piled at Anchorage and the operations curtailed. The Eska Creek coal, which is a high-volatile coal, as loaded carries 20 to 25 per cent. ash, and therefore is not a high-quality coal, though it burns well enough to use on locomotives. The high ash is inherent but can be lessened by washing. The Chickaloon coal is naturally of high grade, but is interbedded with impurity, from which it must be freed for Navy use.

About 10 years ago, the Navy requested an appropriation for investigating coals in Alaska. At that time, attention having been attracted to the Bering River field, an expedition in charge of R. Y. Williams, an experienced mining engineer of the Bureau of Mines, was sent to get out a 900-ton sample for the Navy. The first development started on what is called a 40-ft. bed in the Cunningham tract, which had entered largely into the discussions in Congress, but within 300 ft. the bed thinned out to nothing and its continuation could not be traced. The balance of the sample was then mined from other beds, and after recleaning by the Navy was tested on one of the Government ships; it was found to be good fuel, but not as good as had been expected. Later another appropriation was made for investigating the coal in the Matanuska field, Chickaloon being decided as the most likely place. So about 900 tons of picked coal was mined and transported by sleds to the coast and tested on a Government steamship. This proved, by both analysis and actual trial, to be a trifle better than the average New River and Pocahontas coal used by the Navy; but it must be borne in mind that this was a hand-picked coal.

The natural conditions in the areas of coal metamorphosed to the bituminous and semi-anthracite stages are extremely difficult for mining operations. The reports submitted weekly of the Matanuska Field government operations show this situation, particularly in the higher grade coal area at Chickaloon. There the original development started in the outcrop of the coal bed, which was 12 or 15 ft. thick, and dipping about 45 degrees. Numerous crosscut tunnels were made drifting along the several beds, but these proved very uncertain. In following the larger bed down the dip, the dip changed rapidly to vertical and beyond, but at about 300 ft. resumed approximately its former dip; the coal thinned to 4 or 5 ft. thick to the dip and also thinned along the strike in each direction.

Unquestionably there is a large amount of coal in the aggregate, but it is going to be difficult to mine unless some more undisturbed area is found. There are also the complications, as Mr. Evans and Mr. Moore have pointed out, from igneous intrusions.

From a strategic point of view, however, it is highly important to the country to determine whether or not there is in Alaska coal that will meet naval requirements in sufficient quantity, which can be mined at a

cost approximating that of New River and Pocahontas, plus the cost of delivery to the Pacific Coast. As the uncertainties of the coal and certain limitations of the Leasing Act tended to prevent concerns from taking a chance, it seems proper that Congress should have appropriated funds ($1,000,000) to the Navy to prospect and develop the coal sufficiently to determine whether there was workable coal, and it must be looked at very differently from the situation in the New River and Pocahontas fields, because of its importance to the Navy.

It now appears to be a question of quantity of coal of workable thickness rather than quality, as tests conducted at the Bureau of Mines Station, at Seattle, show that Chickaloon coal, when washed, is of good quality. Although the geologic movements have crushed the coal and mixed in impurities from partings and walls, it is low in sulfur and inherent ash; it may be freed from the shale inclusions by washing. If briquetted it would make a suitable coal.

H. Emerson, New York N. Y.—Twenty years ago I opened a coal mine in Alaska and tried to operate it. I came near being indicted and put into jail by the Government. I then came to the conclusion that you could divide the Alaskan mines into three categories—those in which there was an abundance of coal, not worth while to mine; those that had coal that could be shipped but with veins so faulty you could not mine it; and those where there was exceedingly good coal but no possibility of shipping it, because the sea was too shallow for any boats to get in. I visited Alaska last summer and found that its development is not as far along as it was twenty years ago.

Geology of the Namma Coal Field, Burma

By Edel Moldenke, E. M., Watchung, N. J.

(Wilkes-Barre Meeting, September, 1921)

Burma has long been known for its ruby, tungsten, and tin deposits, and, lately, for having the largest lead-zinc mine in the world, the Bawdwin mine of the Burma Corpn. All the coal used, however, is imported from Calcutta; this is a bituminous coal of about 11,000 B.t.u., 24 per

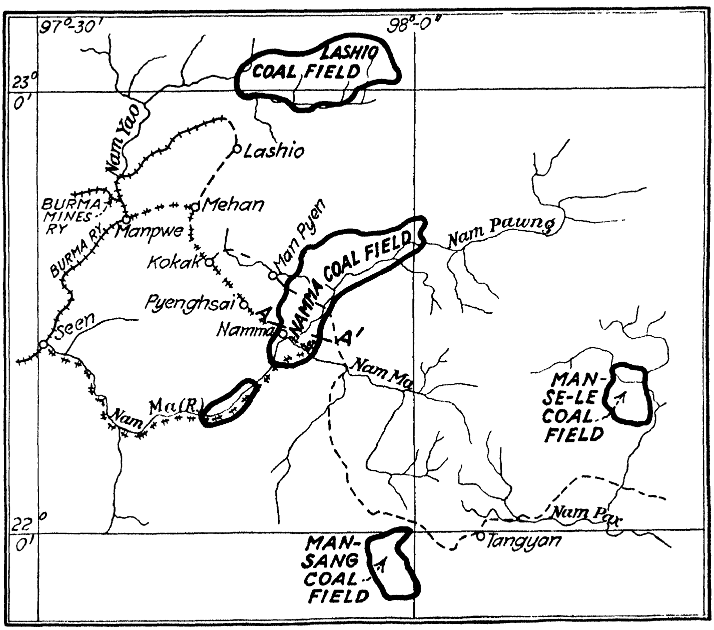

Fig. 1.—Map showing location of coal fields and transportation routes.

cent. ash, and costs about $8 per ton. Bituminous coal has not, thus far, been found, but many lignite deposits are known. The most promising lignite deposits are in the Namma field, Fig. 1, situated in the Northern Shan States of Upper Burma, about 30 mi. south of Lashio, the terminus of the Burma Railway, 750 mi. north of Rangoon. The

most promising part of the basin is about 11 mi. (17.7 km.) long and 5 mi. (8 km.) wide, with an area of 50 sq. mi. (129.5 sq. km.).

The field was first explored in 1891. In 1906 a few test pits were put down and an unsuccessful locomotive test was made by the Burma Railway from coal taken near the surface. In 1918–19, a careful exploration made by the writer, for the Burma Mines, Ltd., resulted in proving considerable of the bed, which was opened in 1919 by a double-track slope to a depth of 500 ft. (152 m.); it has since been extended to 750 ft. (228.6 m.). The outcrops are few in the gently sloping country, with about 10 ft. (3 m.) of soil cover; but sufficient outcrops were found along the main stream of the Namma River and on small side streams to prove the location of the coal.

The basic rock of the coal field is limestone of Devonian age, the coal being found in overlying Tertiary beds; the limestone is gently folded but

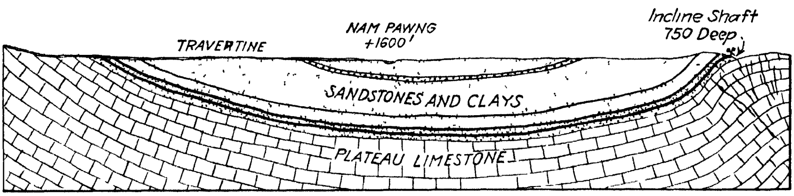

FIG. 2.—SECTION AA′ OF NAMMA COAL FIELD.

badly shattered in some places. A cross-section of the measures, Fig. 2, on the line *A-A′* of Fig. 1, indicates the probable general outline of the basin, in which there appear to be no serious faults, although there are minor structural features near the surface.

The columnar section of measures, Fig. 3, shows two main beds: the upper one from 10 to 15 ft. (3 to 4.6 m.) thick with an average of 12 ft., has a characteristic parting about the middle from 3 to 6 in. (7.6 to 15 cm.) thick; the lower bed ranges from 19 to 40 ft. (5.8 to 12.2 m.), is of uniform character from top to bottom, and has an average thickness of about 21 ft. (6.4 m.) in the area examined. The stratum between the two beds is loosely cemented sandstones and clays, and maintains a thickness of 21 ft. Below the lower bed is a layer of quicksand 2 ft. (0.6 m.) thick, which is heavily charged with water.

The coal is badly weathered at the surface, but below surface influence it is a lustrous, bright, black lignite with dark brown streak, having a specific gravity of 1.4 to 1.5. The coal breaks with a conchoidal fracture. On exposure, it loses some moisture and breaks into cubical fragments. It burns with a bright flame and strong odor and yields a light coke of no strength; an analysis will average: water, 14 per cent.; fixed carbon 45 per cent.; volatile matter, 37 per cent.; ash, 4 per cent.; B.t.u., 9000.

The writer considers 50,000,000 tons of coal in the lower and 30,000,000 in the upper bed a very conservative estimate of the contents of the

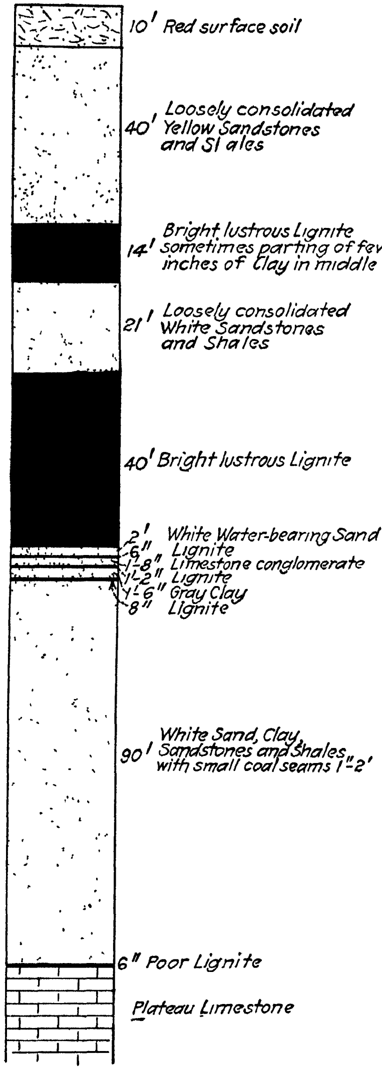

FIG. 3.—COLUMNAR SECTION TAKEN NEAR INCLINE.

proved portion of the basin. As no mining has been done, the possible percentage of recovery has not been determined.

Two railway extensions from the Burma Railway are possible, each about 35 mi. (56 km.) in length, one from Seen by the Namma River, and the other from Manpwe via Mehan and Kokak, both of which are believed to be practicable. Transportation by rafts on the Namma River would be practicable in the dry season for small outputs.

The mines would probably be worked by Chinese labor from Yunnan, as the local inhabitants, the Shans, fail to see the necessity for hard labor, and do not form a reliable working force.

Due to the low efficiency of labor and the costly transportation, the cost of coal will be high, but, it is believed, considerably less than that of imported coal.

New Features in Structural Geology of the Anthracite Basins

By James F Kemp,* New York, N Y

(Wilkes-Barre Meeting, September, 1921)

In earlier years, the custom prevailed of regarding the anthracite basins as cases of folding with slight development of faulting. Folding is so pronounced and, in the eastern and western Middle Fields, at times so violent, that the inference was a natural one and led to the widespread impression that practically all the disturbances of the seams could be explained in this way. The cross-sections and maps prepared forty or more years ago for the Second Geological Survey of Pennsylvania, by Mr. Ashburner under Prof. J. P. Lesley, exhibit little else than folds.[1] So far as I can discover faults fail in the charts, except in one small case in the Henry colliery near Plainsville on the outskirts of Wilkes-Barre.

Some years ago, while spending a day or two in Wilkes-Barr, under the guidance of R. V. Norris, I looked over some recent and accurate cross-sections of a number of the large collieries and was deeply impressed with the peculiar behavior of some of the seams and the intervening shales. Thus, a seam might present a small, more or less overturned fold, rising 30 to 50 ft. (9 to 15m.) above its general course, and yet the movement would be taken up in the overlying shales, so as hardly to be noticeable in the seams next above or below. These and similar anomalies, especially when shown with the accuracy of engineer's surveys, possess great scientific and practical interest, bearing as they do, not alone on coal mining, but on theoretical geology and the actual behavior of strata under great compressive stresses. In preparation for the semicentennial meeting of the Institute, an effort was therefore made to secure some of the most interesting sections for presentation to the members. The collection of illustrations here shown is mainly due to Douglas Bunting of the local committee, to whom and to the contributing companies acknowledgments are due.

The illustrations are taken from all four of the main basins, but are chiefly from the portions nearest Wilkes-Barre. The first are in the

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* Professor of Geology, Columbia University.

[1] There are several volumes on the anthracite areas, listed as A–2 and A. A. The maps and cross-sections are folded in accompanying pockets, but also are to be found unfolded in the portfolios of the Grand Atlas, Division II, Parts 1 and 2.

southern part of the Northern, or Wyoming, basin. The general geological section is as follows:

	ANTHRACITE REGION	WESTERN PENNSYLVANIA
PERMIAN		*Dunkard* or Upper Barren Series
PENNSYLVANIAN or CARBONIFEROUS	*Coal Measures* thickest in southern or Schuylkill Basin with about 20 coal seams. *Pottsville* Conglomerate thickest in Schuylkill basin, with 6 coal seams.	*Monongahela* or Upper Productive Series, 6 coal seams *Conemaugh* or Lower Barren series. *Allegheny* or Lower Productive series, 7 coal seams. *Pottsville* Conglomerate.
MISSISSIPPIAN or LOWER CARBONIFEROUS	*Mauch Chunk* red shales. *Pocono* sandstone.	*Mauch Chunk* red shales containing also the *Mauch Chunk* limestone. *Pocono* sandstone.

The subdivisions of western Pennsylvania cannot be easily carried out in the anthracite basins, and correlation of the strata must be made chiefly by comparative study of the fossil floras of the seams, as has been in progress by David White. A stratigraphic feature of much importance relates to the present, or former more extensive, existence of Permian strata in the anthracite areas, because on this question and the determination of the section that has been removed by erosion, depends the amount of load carried by the anthracite coal measures when they were folded. The folds and overthrust faults are characteristic of relatively small load. Moreover, individual small folds often become larger and larger as we approach the surface from the deep workings, just as if, with diminution of load, more extensive movements and adjustments became possible.

NORTHERN, OR WYOMING, FIELD

The first section, Fig. 1, is taken at the Auchincloss colliery on the southeast side of the Wyoming basin. There are 17 or 18 seams in the section, the number depending on the splitting into two of what would otherwise be a single seam. The names of the seams from the surface downwards are as follows: Nos. 6 to 10 inclusive; Top George; Bottom George; Abbott; Mills; Hillman; Baltimore; Top Forge; Forge; Top Twin; Bottom Twin, which splits into two on the left; Ross; Red Ash. With some local variation these names will be found on the other sections from this field; the Baltimore seam is generally believed to be the correlative of the Mammoth in the other three fields. On the right-hand

side, the tendency of a fold, small in the lowest seam, to expand to greater proportions above is well illustrated, and the same tendency is shown in the large central anticline, which is overturned below, but grows broader and less violent above. The bulging of the shales and shaly sandstones and their tendency, by adjustment, to take up the displacement is well brought out. The splitting of the Bottom Twin seam and the double names for the George, the Forge, and the Twin, show the difficulty of correlating seams, even in adjacent properties. Unless actual workings are connected, uncertainties often arise. The thickening and thinning of the Baltimore seam are very apparent.

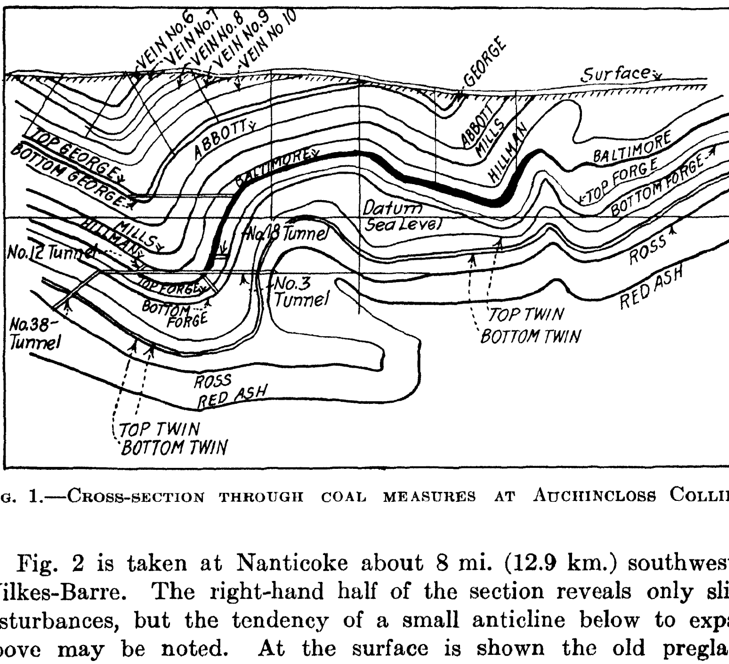

FIG. 1.—CROSS-SECTION THROUGH COAL MEASURES AT AUCHINCLOSS COLLIERY.

Fig. 2 is taken at Nanticoke about 8 mi. (12.9 km.) southwest of Wilkes-Barre. The right-hand half of the section reveals only slight disturbances, but the tendency of a small anticline below to expand above may be noted. At the surface is shown the old preglacial channel, which was buried in water-bearing sands and gravels and the unexpected presence of which led to the sudden rush of water, sand and boulders into the workings in the Ross seam, Dec. 18, 1885.[2]

In the left-hand portion are two small overturns with steep dips facing to the left, which can be traced upward through several

[2] Charles A. Ashburner: The Geological Relations of the Nanticoke Disaster. *Trans.* (1887) **15**, 629.

J. F. Kemp: Buried River Channels of the Northeastern States. *Proc.* Wyoming Historical and Geological Society (1915) **14**, 35–54.

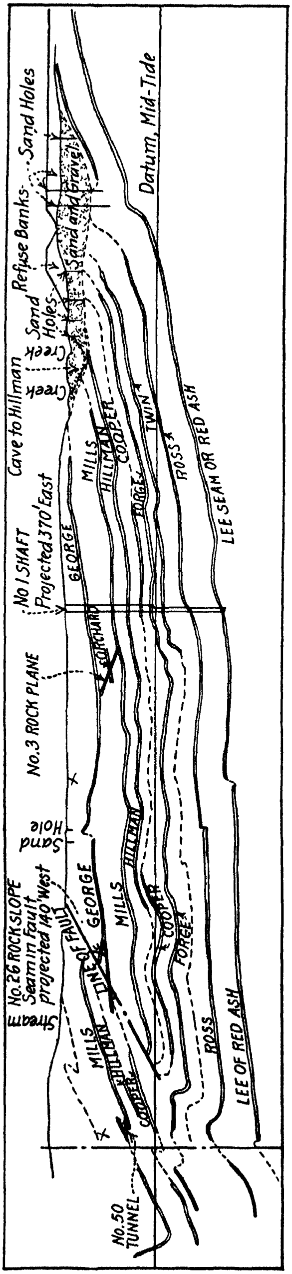

Fig. 2.—Cross-section through Nanticoke property of Susquehanna Collieries Co.

seams, from right below to left above, at angles of 15° to 30°. The great point of interest, however, is the thrust-fold at the extreme left, which, beginning as a comparatively small overturn in the lower seams, develops at the horizon of the Cooper (the local name apparently for the Baltimore) into a strong thrust fault, with a displacement on the 25° dip, of 600 ft. (182 m.). The Forge seam 75 to 100 ft. (22.9 to 30.5 m.) below is apparently not affected; and above it, the compressive strain was taken up in the adjustment of the shales with the great fault displacement mentioned. A second sympathetic overturn is shown still farther to the left, but with small breaks. The opposing dips of the axial planes of the folds and fault last described to those of the small ones farther to the right indicate that the two sets were produced by successive thrusts at different times.

Fig. 3 is part of a much longer section of the measures near Plymouth, in the collieries of the Delaware & Hudson R. R. At the right-hand side, a small thrust fault with a dip of 12° is shown. The main thrust fault farther to the left has a dip of 15° and produces in the Lance, the uppermost seam of the section, a displacement on the dip of 150 ft. (45. m.). A displacement of 240 ft. has been suffered by the five-foot seam, which is 235 ft., vertically lower in the section; whereas the Bennett, 130 ft. vertically below the five-foot, has been displaced only 65 ft. Bulging and adjustment of shales are responsible for these differences.

In the section at the Bliss Colliery, Fig. 4, at the right-hand side, the Red Ash and the Ross seams are affected by an overthrust N-shaped fold with

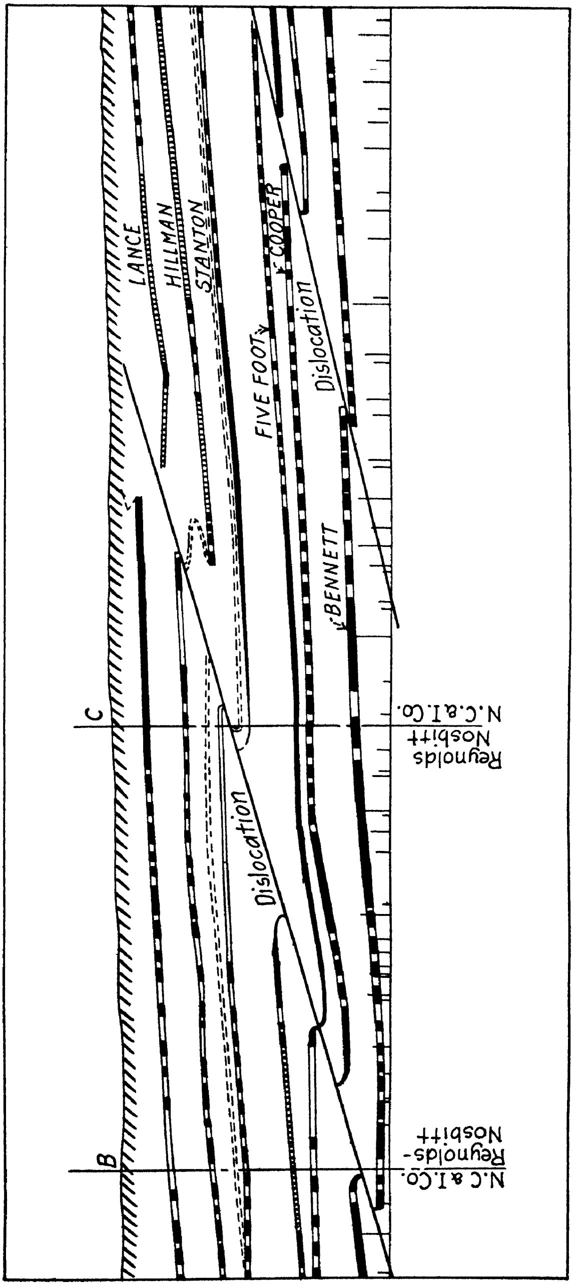

FIG. 3.—PART OF SECTION OF DELAWARE & HUDSON R R. COLLIERIES NEAR PLYMOUTH.

a very steep dip of the axial plane to the left. The bulging of the shales has been such that a sharp trough in the upper, or Ross, seam is brought directly over a sharp arch in the lower Red Ash. If a winze had been sunk from the bottom of the Ross seam trough in the natural expectation of finding a corresponding trough in the Red Ash below, it would have met an arch at less than half the normal interval! Farther to the left are two small overturns. The first is marked in the lower three seams and then is lost in a larger gentler fold above. The second is relatively small in depth, but with decreasing load at the time of folding, becomes larger and larger as we go toward the present surface.

In the section at the Priscilla Lee basin of the West End Coal Co., Fig. 5, an extremely interesting instance of a greatly eroded anticline and accompanying thrust fault is shown. On the right, a drill hole at the end of the "No. 9 Projected Tunnel" revealed red shale with 3 ft. of shelly coal. This would imply that the Mauch Chunk shale, with a

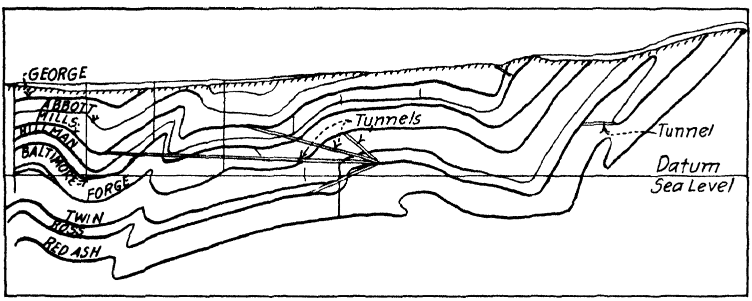

FIG. 4 —CROSS-SECTION THROUGH COAL MEASURES AT BLISS COLLIERY.

dragged-in block of sheared coal was faulted up opposite the Coal Measures. Yet the actual mining had shown the Bottom Red Ash seam 300 or 400 ft. immediately below. The Mauch Chunk shale must thus lie ahead of these workings. Finally, the existence of a great thrust fault is further demonstrated and its displacement amounting to 930 ft. is proved because the upward-thrust part of the Bottom Red Ash seam has been developed near the surface. After this upfaulted portion of the Bottom Ash seam had been traced to the left, it ran out to daylight because of erosion, but still farther to the left it was found preserved with the three next overlying seams in a small, outlying syncline.

In Fig. 6, additional details of the great thrust fault are brought out, and the excessive crumpling of the coal seams in the hanging wall close to the fault is illustrated. Farther to the left, the flat and undisturbed attitude of the Bottom Red Ash seam is one of the impressive structural features of the section. The strain was eased by a limited localized readjustment.

The northern, or Wyoming, Field from which these six illustrations have been taken is, on the whole, the least violently folded of the four main fields. The sections, however, make apparent that it has suffered from severe compression and that the easing of the strain has at times involved thrust faulting of great extent.

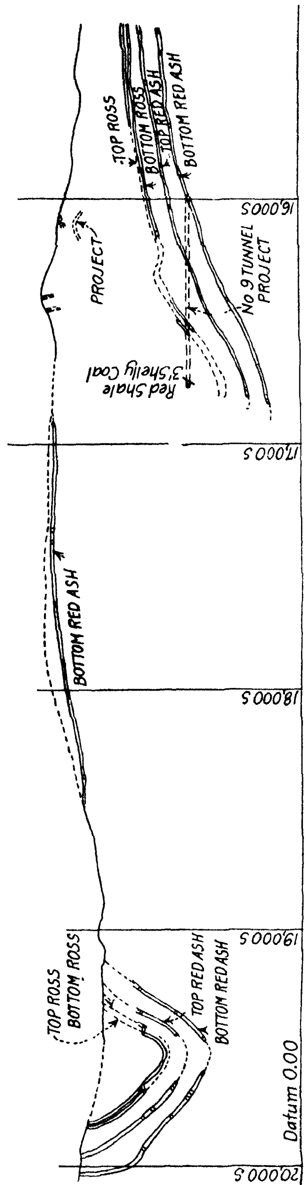

FIG. 5.—SECTION THROUGH DUPONT DRIFT AND PRISCILLA LEE BASIN WORKINGS OF THE WEST END COAL CO.

EASTERN MIDDLE FIELD

Fig. 7 is a section at Audenried. In this field, the local names for the seams are different from those applied near Wilkes-Barre. From the surface downward they are as follows: The Mammoth, which is a double seam, Wharton, Gamma, Buck Mountain, and Lykens. In the central part of the figure, despite the general, apparently gentle, folding, the Lykens seam is upthrust until it is opposite the Wharton, having moved upwards 500 to 600 ft. on the plane of the thrust.

Fig. 8 shows not only a remarkable folded structure in the foreground, with its sharp syncline in the center and small roll on the right, but also, in the background near the top, a remarkable fault that must have involved an extended shift nearly parallel with the strike. In the pit in the foreground with the water, the coal was in place on both sides of the syncline. The coal was also in place on the right-hand side of the syncline, when followed into the extension of the strippings in the background; but on the left-hand side, where the straight bare wall of rock appears, the coal was missing.

WESTERN MIDDLE FIELD

In this field there is another change in the names locally applied to the seams. In the strikingly symmetrical syncline on the right in Fig. 9

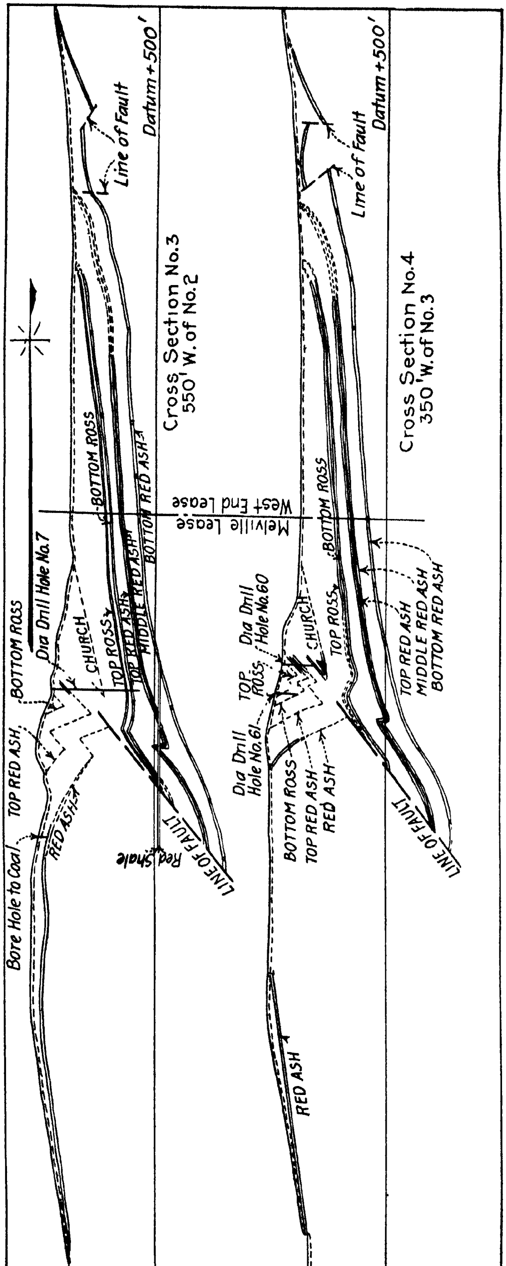

FIG. 6.—CROSS-SECTIONS OF COAL FIELDS, DUPONT BASIN, LOOKING WEST, WEST END COLLIERY.

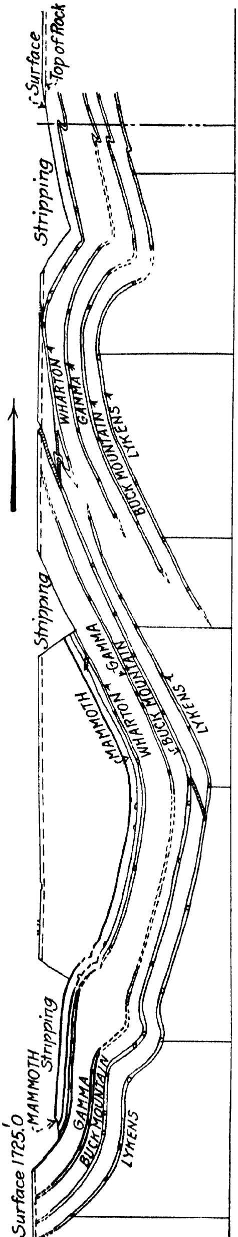

FIG. 7.—SECTION ON LINE OF NO. 11 SLOPE, AUDENRIED NO. 4 COLLIERY.

fifteen seams are shown; their names, from the surface downwards, are as follows: Interval to surface, 200 ft., Spohn; interval 30 ft., Little Tracy; interval 70 ft., Big Tracy; interval 40 ft., Diamond; interval 50 ft., Orchard; interval 135 ft., Primrose; interval 45 ft., Little Primrose; interval 75 ft., Holmes; interval 35 ft., Four-foot; interval 120 ft., top split of Mammoth; interval 100 ft., bottom split of Mammoth; interval 45 ft., Skidmore; interval 65 ft., Seven-foot; interval 80 ft., Buck Mountain; interval 50 ft., Little Buck Mountain. These intervals vary in different places but the figures give a good working idea of the section of about 1100 ft. appearing in the figure. In the syncline the Mammoth seam has two splits about 100 ft. apart. The Top Split scales on the large section 18 ft. of coal; the Bottom Split 26 ft. As we follow them to the left-hand portion of the section they almost unite; then part and finally almost reunite.

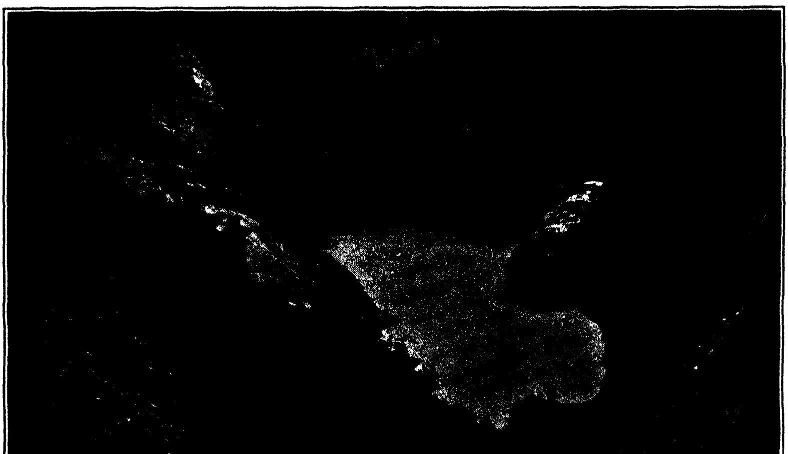

FIG. 8.—PART OF OLD HOLLYWOOD STRIPPING, ONE OF THE FIRST IN ANTHRACITE REGION. THIS STRIPPING IS 300 FT. DEEP.

As the large syncline on the right rolls over into its complementary anticline, a thrust fault has displaced the Buck Mountain seam about 550 ft.; but the overlying Seven-foot extends farther to the left and stops opposite the Primrose. In the adjustments of the shales under the compressive strains, the interval between the bottom and top splits of the Mammoth seam is greatly increased on the hanging-wall side of the fault. To the left of the fault, the two splits of the Mammoth and the Skidmore are affected by a slight thrust fault, which is not shown in the Seven-foot, next below. Still farther to the left, the two splits of the Mammoth are thrust-faulted about 80 ft. without affecting the Skidmore, next below. Farther to the left, a great thrust fault involves a displace-

ment of about 1000 ft. in the Buck Mountain seam, the higher seams having been eroded from the hanging-wall portion.

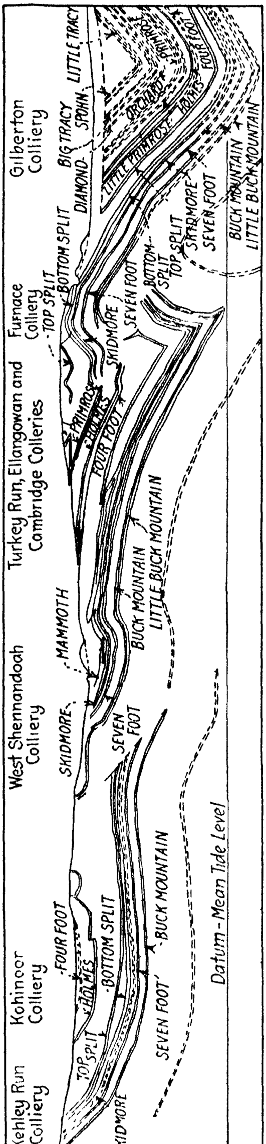

FIG. 9.—SHENANDOAH AND MAHANOY BASINS.

In Fig. 10, a sharp localized thrust is shown on the left-hand side of the syncline, which only produced a Z-shaped roll in the middle and bottom splits of the Mammoth seam, without breaking the continuity, but which displaced, by an overthrust fault of 400 ft., the Skidmore seam 75 ft. lower in the section. Bulging and adjustment of shales have taken up this thrust of 400 ft. without transmitting it appreciably to the top split of the Mammoth.

In Fig. 11, the north limb of a syncline is shown in which an overturned fold has parted in a thrust fault with an unusually steep dip of 50°. While such data as we have indicate that the lower four and relatively thin seams have been rather sharply cut off, the Mammoth seam here about 40 ft. thick, has been turned and dragged up the foot wall of the fault for 200 ft. Above the Mammoth seam on the foot-wall side of the fault, a huge bulging of the shales has been produced, increasing the normal interval of about 200 ft. between the Mammoth and Holmes seams to fully thrice this amount.

SOUTHERN FIELD

An increased thickness of the geological section, together with sharply compressed, deep synclines, characterizes the Southern Field. The sections shown are at the eastern end. In Fig. 12, viewed as a whole, two somewhat complex synclines are separated by the very sharp, broken, upright anticline in the middle of the diagram. About 950 ft. of strata are involved. The extreme squeezing and forcing upward of the Seven-foot, Mammoth, and Primrose seams are very marked when compared with the

lowermost Buck Mountain. Apparently some rupturing at the crest has also taken place, but exploration has not yet shown whether the two ends of the Mammoth seam will join or not. In the next subsidiary anticline on the left, there is an equally marked dragging upward of the

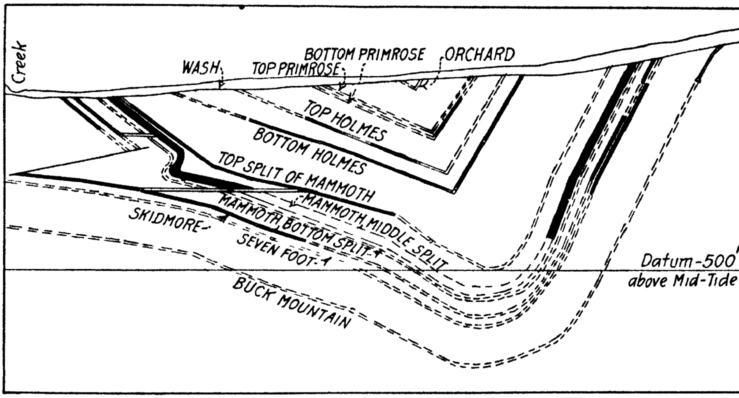

FIG. 10.—CROSS-SECTION OF COAL MEASURES AT SAYRE COLLIERY, MT. CARMEL, NORTHUMBERLAND CO.

Holmes seam, whereas the Mammoth is but slightly affected. In both cases much adjustment has taken place in the associated sediments. In the first case, the Seven-foot seam is pinched and dragged nearly 700 ft. above the Buck Mountain, and in the second the Holmes has

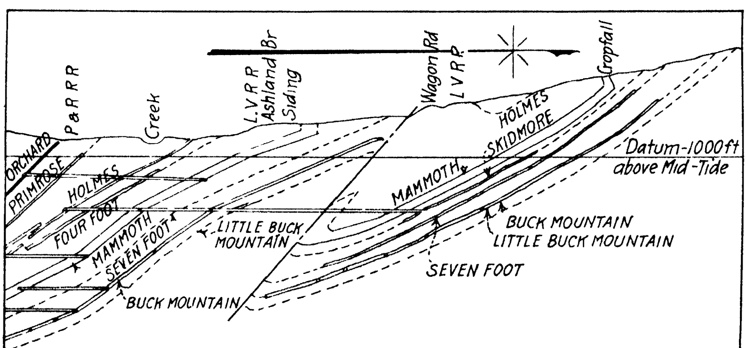

FIG. 11.—CROSS-SECTION OF COAL MEASURES AT PACKER NO. 4 COLLIERY, LOST CREEK, SCHUYLKILL CO.

reached a point nearly 800 ft. above the lower or main split of the Mammoth. The two intervals in the normal section seem to be about 125 ft. and 75 ft., respectively.

In Fig. 13, there are three interesting points. The small tight syncline

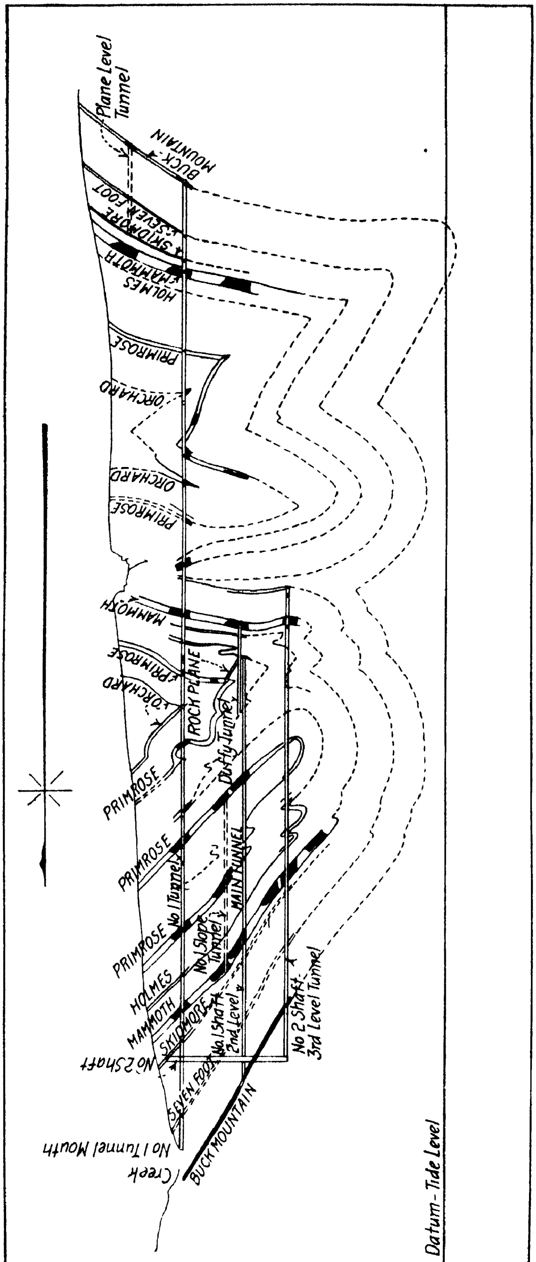

FIG. 12.—CROSS-SECTION K-K, NESQUEHONING TUNNEL.

in the upper right-hand marks the survival after erosion of a sharply compressed extension of the main trough. In the latter, the peculiar roll of the Mammoth seam is worthy of study. At the left, a small and quite independent roll in the Buck Mountain seam has increased the interval between it and the Seven-foot from 160 ft. to 360 ft. The constant surprises which local structures of this sort bring to the mining engineer may well be remarked in passing.

In Fig. 14, a very peculiar little thrust fold with a doubling of the Mammoth seam on itself is shown on the extreme right. This little tight fold extends over 360 ft. of vertical distance. On the lowest tunnel, 600 ft. north (or to the left) a small fold is shown in the Diamond, Little Diamond, Tracy and a very slight fold in the Spohn seam. Small as it appears in the illustration, this fold has increased the interval between

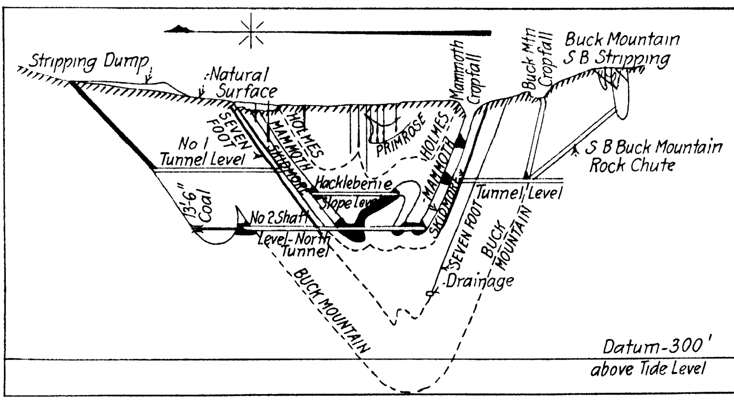

FIG. 13.—CROSS-SECTION D–D, NESQUEHONING TUNNEL.

the Orchard and the Diamond from about 130 ft. on the tunnel above to about 225 ft. in the third tunnel. The most interesting feature, however, is the pinching and stretching of the Mammoth seam in the last anticline on the left. After it was pinched together, the Mammoth seam was stretched up vertically over 600 ft. In the complementary syncline on the left, it is pinched and stretched downward nearly 400 ft. There are other peculiar features in the molding and thickening of the Mammoth seam in the northern limb.

In Fig. 15, about 650 ft. of strata are involved with seven seams. A remarkable anticlinal bulge and thickening with upward stretching of nearly 600 ft. have been produced in the Mammoth seam, and have influenced the four above it, but in so far as the inferred position of the two below are concerned, they are but slightly affected.

In the oral presentation of this paper, additional sections were shown as greater latitude was possible with lantern slides than is the case with

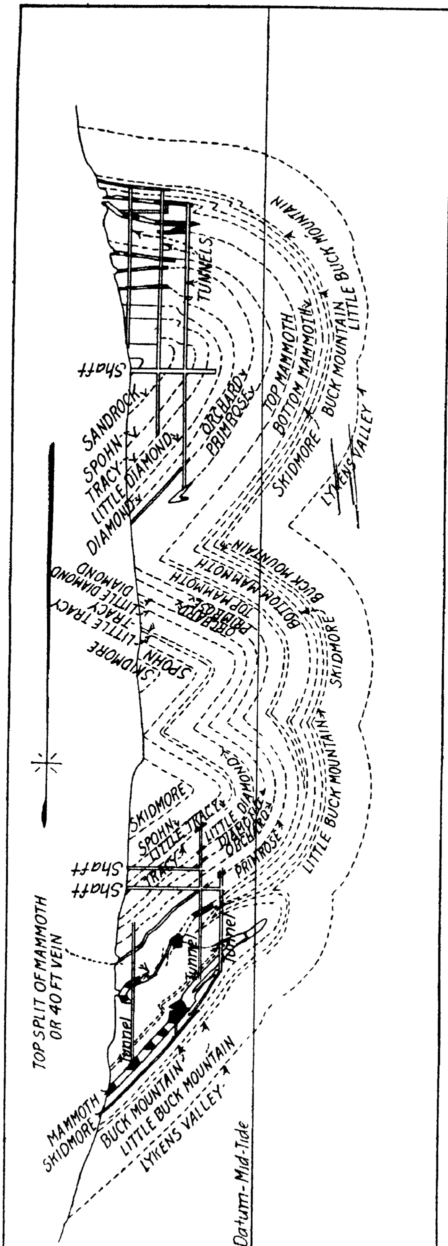

FIG. 14.—CROSS-SECTION THROUGH PANTHER CREEK VALLEY.

the engraved figures. But the ones reproduced here serve to bring out the main features. The existence of thrust faults of a displacement to be measured on the dip of as much as 1000 ft. is the most important structural feature brought out. The tightly compressed and stretched folds, although more generally familiar, can never fail to be an object of interesting study to the geologist concerned with mountain-making

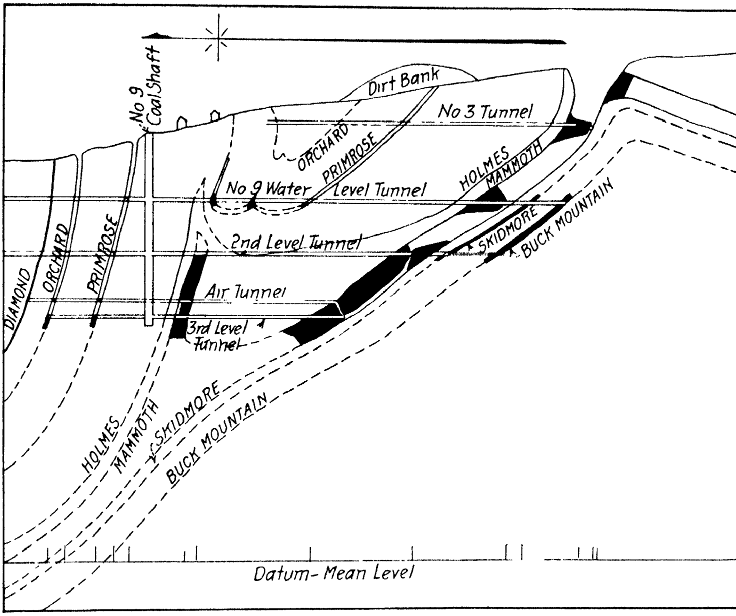

FIG. 15.—CROSS-SECTION ON LINE OF NO. 9 WATER-LEVEL TUNNEL, COAL DALE COLLIERY.

processes. Where they can be illustrated from the engineers' surveys of the mines, they have an exact quantitative character of extreme value to the investigator. Some of the features commented upon, cannot fail to suggest the conclusion that the anthracite measures did not rest under a great load when they were folded, but that they were fairly free to break and slip by, when strained beyond a moderate ability to resist.

Mine Fires Extinguished by Sealing

By Douglas Bunting,* M. E., Wilkes-Barre, Pa.

(Wilkes-Barre Meeting, September, 1921)

In the anthracite fields of Pennsylvania, mine fires occur with more or less regularity and their existence is an ever-present hazard in coal mining. In all probability 90 per cent. of the mine fires can be ascribed directly or indirectly to the ordinary miner's open lamp. Other causes may be smoking, electrical installations, gas explosions, gas feeders, and the communication of fire to the outcrops from ash dumps, culm banks, timber and brush fires.

Fires in pump rooms, engine rooms, and barns have been largely eliminated as a result of the Act of Assembly, approved June 15, 1911, which provides that all inside buildings, including engine rooms, pump rooms, barns, etc., shall be constructed of incombustible materials. Smoking of pipes, cigars, and cigarettes should be prohibited in the mines and too much care cannot be exercised in preventing fires by enforcing all rules designed for that purpose. Mine fires that involve underground fighting are invariably dangerous, and all fires, irrespective of the method pursued in extinguishing them, involve property loss and expense.

Each fire generally presents a more or less original problem, dependent on the particular conditions. Generally mine fires may be divided into two classes: Those occurring in inaccessible workings and those occurring in accessible workings. The methods pursued in extinguishing or controlling mine fires are as follows: (1) direct attack, (2) smothering by sealing, (3) flushing with silt or other solids, (4) flooding with water, (5) cutting off, (6) surrounding with incombustible materials, (7) digging out by stripping from the surface.

Most fires are extinguished in their incipiency by direct attack and failure to extinguish by this method is due to late discovery, lack of water, and facilities for fighting, or improper procedure in the initial attack. Should the method first pursued be unsuccessful, careful consideration should be given in determining next the method to be adopted; then the work should be carried on as rapidly as possible and under the supervision of the most competent persons. Materials and supplies should be furnished without delay and in sufficient quantities for the work to be accomplished.

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* General Superintendent, Lehigh & Wilkes-Barre Coal Co.

Following the discovery of fire, the foreman or superintendent should organize the men available at the location of the fire and, if necessary, send for others experienced in fire fighting. The regulation of the ventilation should be promptly attended to by the foreman or his qualified assistants. The regulation of the air supply to the fire area is of prime importance as too much air will generally increase the extent of fire and too little will increase the liability of explosion.

As most fires occur in accessible locations, they are naturally fought direct in their incipiency and in most cases are so extinguished. But the extinguishing of fires by direct attack has not been confined to small fires or fires in their incipiency as fires have been extinguished by this method after months of work. However, the direct fighting of fires over long periods of time is falling into disfavor and the safer and more efficient method of sealing where practicable is generally practiced.

The speed of travel of a mine fire is dependent on the conditions in the immediate vicinity of the fire. With dry timber and a strong ventilating current, a fire will make rapid progress; while in coal, the progress is extremely slow, except where there is communication with an overlying vein through caves and broken ground.

The most practical, and probably the most efficient, means of direct fighting is with water directed in hose streams of good pressure and capacity, which can only be secured by providing pipe lines of adequate capacity. A system of water pipes, standardized as far as possible, should be maintained for immediate use in mines liable to fires, and maps giving the location of these lines should be provided periodically to the colliery officials. Pipe, hose, connections, valves, wrenches, nozzles, etc. should be maintained at definite points of convenience.

In the sealing of fires, the conditions favorable to this method are: limited area of workings to be sealed and minimum number of seals required. The unfavorable conditions are extensive workings and caved workings with liability of connections through cracks and crevices to overlying veins or the surface. The principal danger is the liability of explosion, which prompts the question as to whether the intake or return stoppings should be erected first. The question has always been a matter of dispute; in 1912, the editors of *Coal Age* invited discussion which resulted in thirty-three opinions being given. Of these sixteen would close the intake first, ten would close the return, six would close the intake and the return simultaneously, and one would close either intake or return first.

There have been many failures to extinguish fires by flooding the areas with water because anticlines and pockets in the workings prevented the water submerging the fire, owing to the compression of the mine air. To illustrate the sealing method of extinguishing mine fires the procedure followed in extinguishing two fires by this method is given.

South Wilkes-Barre No. 5 Colliery Fire

The fire in the South Wilkes-Barre No. 5 colliery originated on No. 13 slope close to the 3d west gangway at point *X*, Fig. 1. A workman had repaired a roof pulley at this place during the night of Feb. 20, 1919, and it is presumed that the timber was ignited by his lamp about 10 o'clock. It was discovered at 4.45 the next morning by a fire boss, who found several sets of timber on fire. He immediately went to the surface and reported to the inside foreman, while the other fire bosses proceeded to fight the fire with two hose streams of water. When the foreman and superintendent arrived, two sets of timber had burned through and some top rock had fallen. The fighting of the fire with hose streams was continued at this location until 10 o'clock, when it was found that the fire had traveled up the slope to the No. 23 tunnel west gangway, as there was considerable timber while falls of roof made the work of fighting slow. The men were then moved to the No. 23 tunnel west gangway and continued fighting direct with two hose streams until 10 o'clock that night, when it was found that the fire had passed the 2d west gangway. The men were then moved to this gangway and the fire fought from there until the next noon, when it was decided to seal the fire area.

During the 31 hours of direct fighting the ventilation was regulated to give only sufficient air in the immediate vicinity of the fire to avoid accumulation of smoke and gases, which would seriously interfere with the work; also the water supply was augmented by utilizing a compressed-air pipe line from the surface to the No. 23 tunnel east gangway.

The ventilation of these workings, previous to the starting of this fire, is shown by arrows. It will be observed that the intake passed up No. 13 slope and slope airway and returned to the face of the 3d west gangway where it passed through a regulator *R'* located in a rock plane to the Top Baltimore vein. The quantity of air passing near the top of No. 13 slope to No. 18 slope workings was approximately 14,000 cu. ft. per min. and the quantity returning through the regulator *R'* to the Top Baltimore was approximately 20,000 cu. ft. There was a small split of air passing in the 3d west airway and through the regulator *R'*, where it joined the split from the fire area.

The hoisting shaft at this colliery has a depth of 1040 ft. and the distance from the foot of shaft to the origin of the fire is 6500 ft. The origin of the fire is 1300 ft. below the surface.

To seal the fire, all materials were transported on the 3d west gangway from No. 2 slope to the foot of No. 10 plane and to No. 25 tunnel and carried by men from these points to the locations of the seals. The seals were located at points 2, 3, 4, 5 and 6 and were constructed by first securely setting between roof and bottom 6-in. props spaced about 4 ft.

To these props brattice boards were nailed and the stoppings made reasonably tight by fitting small pieces of brattice boards into the irregularities of the ribs and roof and nailing to the brattice boards. The cracks in the brattice were then covered with battons having the top edge beveled to permit sealing, which was done by applying lime mortar to the edges of the battons and along the roof and ribs.

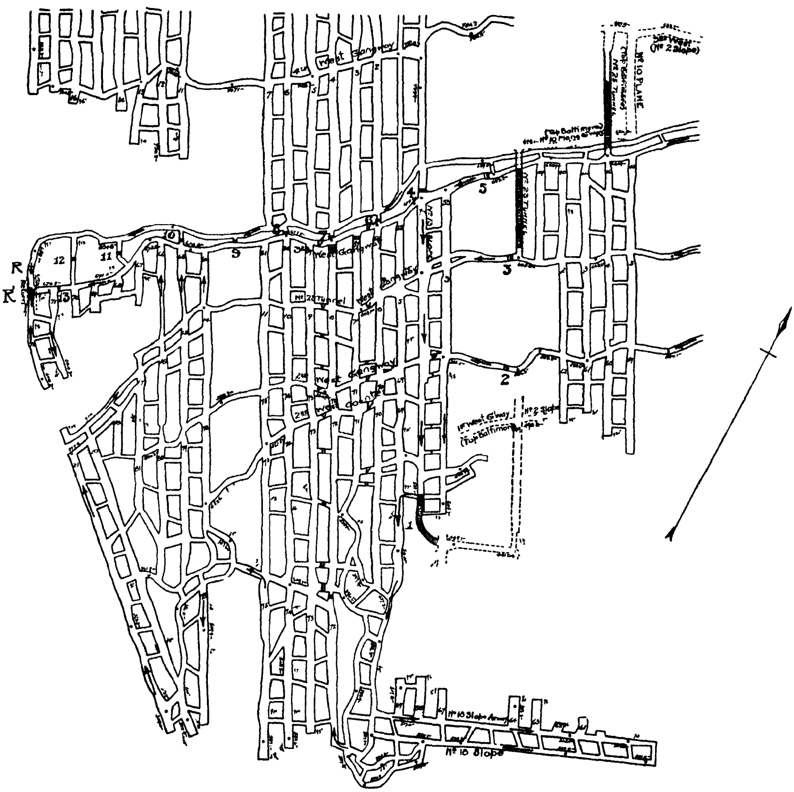

FIG. 1.—PLAN SHOWING LOCATION OF FIRE AND MINE WORKINGS INVOLVED, SOUTH WILKES-BARRE NO. 5 COLLIERY.

Seal 2, on the 2d west gangway, was built with a 5 by 5-ft. opening but seals 3, 4, 5, and 6 were built continuous and without openings. Seals 2, 3, and 5 were completed first and immediately upon completion of seals 4 and 6 a door was put over the 5 by 5 ft. opening in seal 2. This door was made of two thicknesses of brattice boards with canvas between and a sealing strip at bottom. This door was put up at 3.20 A. M., Feb.

23, and all men immediately withdrawn. The fire area was now closed on the intake side as headings 7 to 12 on the 3d west gangway contained old walls. Where No. 13 slope passes through the rock from Bottom Baltimore to Top Baltimore, at location of seal 1, there was an extra main door, which was closed at 7 A. M. Feb. 24, covered with canvas, and banked with dirt along the bottom. A concrete seal 8 in. thick was built against the door and completed the same day. On the following day concrete seals 2, 3, and 5 were built against the brattice stoppings put in on the 22d, and on the 26th concrete seals 4 and 6 were built against the brattice stoppings previously constructed. On the 27th, the work of building concrete seals 7 to 12 against the old walls was begun. After the completion of the seals, the 3d west gangway was closed between chambers 70 and 71 at 4 P. M. March 1, after some difficulty because of escaping gases.

All of the concrete seals, with the exception of No. 13, were built against stoppings previously constructed, by first standing wooden mine rails about 2 ft. apart and approximately 12 in. from the original stopping. Boards were then put on the inside of the vertical wooden mine rails and concrete placed between these and the original stoppings. At the time of putting in these concrete seals, pipes were put in seals 1, 2, 3, and 4 for the purpose of taking samples of the enclosed air for analysis.

The analysis of mine-air samples from seals 1 and 2 are shown graphically in Fig. 2. The dimensions of the seals are given in Table 1. The amount of concrete placed in the thirteen seals was 37 cu. yd.

The sealed area covers 32.8 acres and the volume of the mine openings amounts to approximately 4,350,000 cu. ft. An average section of the vein is 7.75 ft., divided as follows: Coal 2.15 ft., rock 0.70 ft., coal 4.90 ft. A maximum of 79 and an average of 17 men were employed in fighting the fire and building the seals, a total of 4363 man-hours, at a cost of $2555.15. The labor expended from March 3 to 29 was used in going over the seals and pointing all cracks and leaks.

On May 29, the seals were opened by first removing the upper portion of concrete of seal 1, after which an opening 10½ in. high and 27 in. long and about 2 ft. below the roof was made through the door, which formed the backing of this concrete seal. The gases emitted from this opening filled the head of the slope in a very short time. An opening 7 ft. square was then made in seal 13, which rapidly relieved the gas pressure at this point; and finally the 5 by 5-ft. trap door in seal 2 was removed at 3.10 P. M. of the same day. The men employed in this work were then withdrawn. At 7 o'clock the next morning, the superintendent, foreman and several fire bosses examined the fire area and found that the fire had been extinguished; they also found that the fire had extended to the second heading above the 2d west gangway, a distance of 520 ft. from its origin, where it had died out soon after the closing of the intake seals.

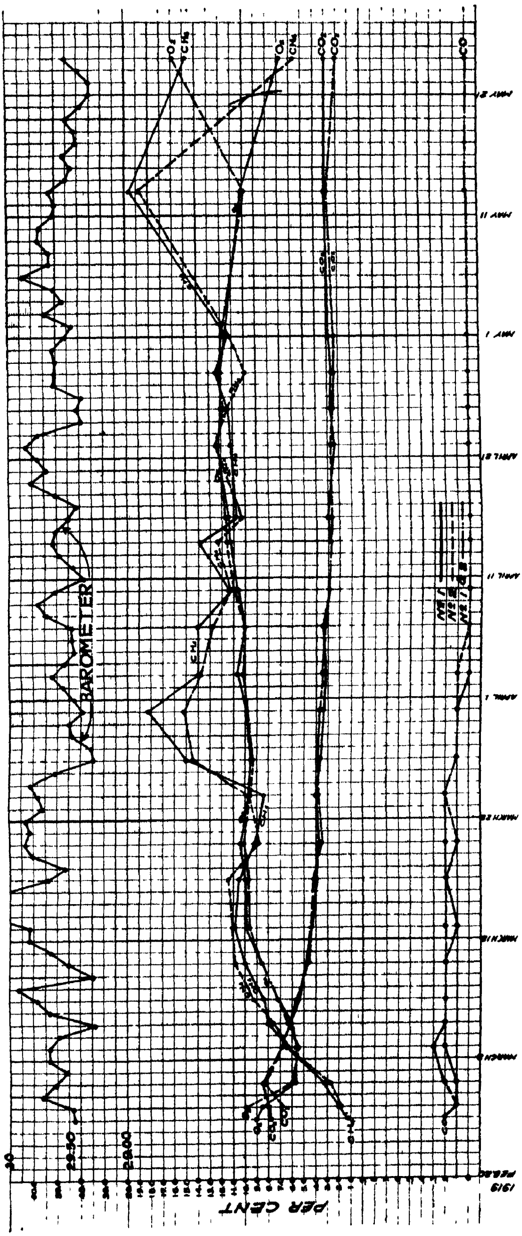

FIG. 2.—COMPOSITION OF MINE AIR IN SEALED AREA; TAKEN THROUGH SEALS NO. 1 AND NO. 2, DURING FIRE IN NO. 13 SLOPE AT WILKES-BARRE COLLIERY FROM FEB. 25 TO MAY 24, 1919.

TABLE 1.—*Dimensions of Seals South Wilkes-Barre No. 5 Colliery*

No.	Length		Height		Thickness Inches	(Volume) Cubic Feet
	Feet	Inches	Feet	Inches		
1	13	4	8	6	8	75.6
2	13	4	7	6	9	75.0
3	15	2	7	0	12	106.2
4	15	3	10	2	12	155.0
5	21	0	8	0	10	140.0
6	11	0	6	7	10	60.3
7	4	10	5	7	10	22.5
8	24	0	6	0	10	120.0
9	9	10	5	5	10	45.0
10	8	0	5	4	10	35 6
11	10	0	4	0	10	33 3
12	9	3	5	0	10	38 6
13	14	0	7	2	11	91 9
						999 0

NOTTINGHAM NO. 15 COLLIERY FIRE

This fire originated opposite chamber 61 on No. 1 basin gangway, 12th east, No. 3 slope at point *X*, Fig. 3. A laborer working in this vicinity at 2.45 P. M., May 1, 1919, went to the miner's box and filled his lamp with oil; he then brushed the lamp wick against the box to spread the cotton and knock off the incrustation, and left immediately so as to catch the man trip up the slope. At 6.20 P. M., a fire boss found the box ablaze, also the platform on which it stood, and some old ties near by. He went for help and on his return found that the fire had reached the brattice and the timber in the chamber. A fall of top coal had occurred during his absence and there was evidence of fire under it.

While the fire was fought with water carried in powder kegs from the ditch on the 13th east gangway, hose was being secured and word sent to the assistant foreman. About 9 o'clock hose had been obtained and a stream was played upon the fire until 10 A. M., May 4, when two additional hose streams were secured by utilizing a 3-in. high-pressure air pipe on the 11th east gangway and extending with 2½-in. pipe and hose down chamber 56 to the 12th east gangway, thence in the gangway to the fire. During the fighting of the fire, which had communicated to the fall of top coal, 41 cars of coal were loaded and removed by 8 A. M., May 3, but in so doing many of the men were overcome by the gases of combustion. At this time the organization fighting the fire was disrupted by trouble in another section of the mine, so the fire gained considerable headway. Fighting continued, however, until 2 P. M., May 4, when it was decided to seal the fire area. During the 65 hr. of direct fighting, the ventilation was

regulated to give sufficient air in the immediate vicinity of the fire to prevent serious accumulations of smoke and gases.

The ventilation of these workings previous to the fire is shown, in Fig. 3, by arrows. It will be observed that the intake entered from the 11th and 12th east gangways and returned on the 12th east airway. The quantity of air in the return was approximately 20,000 cu. ft. per min. The hoisting shaft has a depth of 365 ft. and the distance from the foot

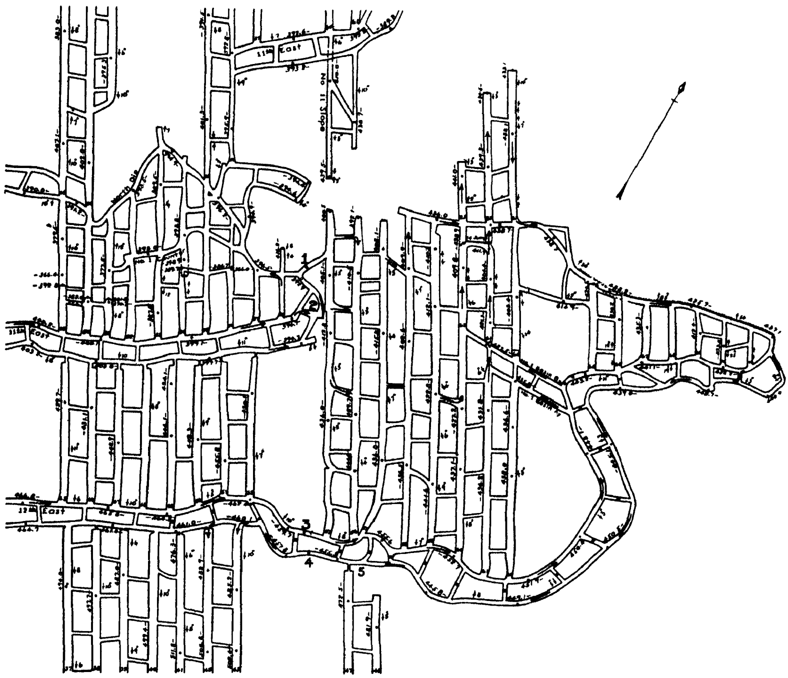

FIG. 3.—PLAN SHOWING LOCATION OF FIRE AND MINE WORKINGS INVOLVED, NOTTINGHAM NO. 15 COLLIERY.

of shaft to the fire was 12,100 ft. The depth from the surface to the origin of the fire is 950 ft. Materials used in the construction of the seals were transported on the 11th and 12th east gangways to within 100 ft. of the seal locations. The locations of the seals are indicated by numbers 1 to 5. They were constructed by setting two rows of wooden mine rails between the roof and bottom, spaced about 3 ft. apart, with from 14 to 26 in. between the props, depending on the thickness of seal to be built. To the inside of these props, sheathing boards were nailed and the space between filled with concrete.

The construction of seals 1 and 2 was started at 2.30 P. M., May 4, and seal 3 at 6.00 P. M., May 5. Seal 1 was built without an opening

but seal 2 had a 4 by 4-ft. opening and seal 3 a 6 by 6-ft. opening. Seal 1 was completed at 2.45 P. M., May 6, and seals 2 and 3 were completed about the same time, except for closing the openings, which were closed with concrete during the next hour. The fire area was thus closed on the intake side and all men immediately withdrawn. Immediately after seal 3 was closed, the doors in the two headings outside of the seal were thrown open, thereby short-circuiting the air to the 12th east airway.

At 9.15 P. M., May 7, after an examination was made, with the use of oxygen helmets, the location for seal 4 was chosen and work started;

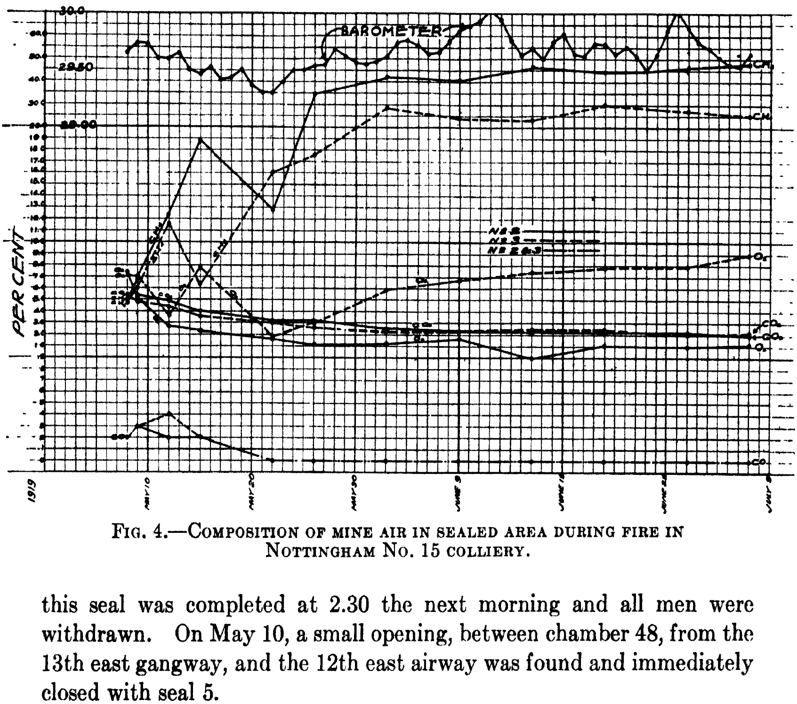

FIG. 4.—COMPOSITION OF MINE AIR IN SEALED AREA DURING FIRE IN NOTTINGHAM NO. 15 COLLIERY.

this seal was completed at 2.30 the next morning and all men were withdrawn. On May 10, a small opening, between chamber 48, from the 13th east gangway, and the 12th east airway was found and immediately closed with seal 5.

When seals 1 to 4 were built, pipes were put in so that samples of the enclosed air could be taken for analysis; also an 8-in. cast-iron pipe, with an elbow to form a trap, was placed in the bottom of seal 3 for the discharge of the water flowing in the ditch of this gangway. The hose line connected with the regular 12th east gangway water line was also arranged so that samples of the mine air could be taken through it from a point as near the fire as possible. The analyses of samples from seals 2 and 3 are shown graphically in Fig. 4.

The dimensions of the seals are given in Table 2. The amount of concrete used in the construction of the five seals was 40.4 cu. yd. The

sealed area covers 24.2 acres and the volume of the mine openings amounts to approximately 5,300,000 cu. ft. An average section of the vein is 21 ft. divided as follows:

	Feet	
Coal	10.00	Top coal
Refuse	1.40	
Slate	1.90	Big slate
Coal	7.30	Bottom coal
Refuse	0.40	
Total	21.00	

A maximum of 110 and an average of 30 men were employed in fighting the fire and building the seals, with a total of 6846 man-hours, at a labor cost of $3979.60.

On July 12, the opening of the seals was begun at 6.45 P. M. by cutting out the concrete of seal 4, and an opening approximately 8 by 10 ft. was made in the inside sheathing of this seal by 11.15 o'clock thereby opening the return. A half hour later, work was started on seal 2 and the concrete in the 4 by 4-ft. opening was cut out by 4 o'clock the next morning. The doors in the headings outside of seal 3 were then closed. Preparatory to opening seal 2, the work of removing the concrete in the 6 by 6-ft. opening in seal 3, was begun; this was completed at 5.15 the following morning.

TABLE 2.—*Dimensions of Seals, Nottingham No. 15 Colliery*

No	Length		Height		Thickness, Inches	Volume, Cubic Feet
	Feet	Inches	Feet	Inches		
1	15	0	16	0	9 to 18	320.1
2	25	4	7	6	24 to 32	455.1
3	17	0	8	6	18	216.7
4	12	0	8	0	12	96.0
5	3	6	3	6	4	4.1
						1092.0

The men employed in this work were then withdrawn and the next day the superintendents and foremen made an examination of the fire area. They found that the fire had been extinguished and also that it had not extended beyond the first heading on the right of chamber 61, a distance of approximately 50 ft. from its origin.

This paper is presented in the hope that discussion of the methods of fire fighting will result and also that it may prompt the submission of other experiences in extinguishing or controlling particular mine fires in the anthracite field.

The writer acknowledges the assistance of J. D. Joseph, R. G. Carpenter, F. W. Seymour, and D. J. Thomas, colliery officials in collecting data relative to these two fires and also extends his thanks to M. S. Hachita for the preparation of tables and diagrams of air analyses.

DISCUSSION

George S. Rice,* Washington, D. C.—The U. S. Bureau of Mines has always been deeply interested in mine-fire fighting; but its aid has been sought more in the western metal mines and in the bituminous coal districts than in the anthracite.

While the fires described have been admirably handled, there are a number of interesting features in the charts showing the rise and fall of the component gases behind the seals. Two or three things seem quite at variance with our experience with the gases produced by bituminous-mine fires that have been sealed. First, the oxygen content of a sealed area nearly always falls off rapidly after sealing and keeps falling, sometimes until it reaches zero, if the stoppings are tight. In both fires here described, if I understand the graphs correctly, the oxygen content decreased and then increased, which seems odd.

We have also found that the presence of carbon monoxide is a good indication of an active fire. Usually, the content has been higher at the start than shown by these graphs, in such fires as have been under my observation; then, as the flame has gone down, the carbon-monoxide content has dropped. However, the absence of carbon monoxide does not indicate that the fire is out, only that the flame is extinguished; the hot coals may remain ready to burst into flame when the sealed area is reopened. Therefore, it seems to me that in considering this matter, it would be helpful if Mr. Bunting would explain why the oxygen content apparently increased.

Another question is, what was the indication to the mine management that the fire was out? The time that had elasped in each case seems rather short. We have encountered fires (and I have no doubt you have in the anthracite district) when reopening sealed areas after long periods, even where it was thought the seals were tight. In the cases here discussed, was the coal on fire, or was the fire confined to timber? It has been our experience that when coal gets on fire, the hot coal will often be surrounded by ash and retain the heat for long periods of time—months and perhaps years—if there is a large body of fire. Also, was breathing apparatus used for preliminary exploration before restoring ventilation, to determine if the fire was extinguished?

Douglas Bunting.—No breathing apparatus was used in the South Wilkes-Barre fire. At the Nottingham fire, breathing apparatus was not used until it was necessary to seal the return. In both fires there was burning coal—more in the Nottingham fire where there was but little timber.

The analyses of the South Wilkes-Barre fire show, at seal 1, a decrease and then a slight increase in the oxygen content; the same variation

* Chief Mining Engineer, U. S. Bureau of Mines.

applies generally, to seals 2, 3, and 4. At the Nottingham fire, the oxygen content diminished and then increased at seal 1; at seal 2, there was a decrease; at seal 3 there was a decrease and then an increase.

There was no positive indication that these fires were extinguished prior to opening the seals but our opinion was that the carbon monoxide content was the only indicator. When this was reduced to zero, we thought that the fire was out. In the case of the South Wilkes-Barre fire, the carbon monoxide disappeared in seals 1, 2, and 4, on April 7, and in seal 3, on April 10, so that the disappearance of the carbon monoxide occurred practically at the same time throughout this entire area. At the Nottingham fire, the carbon monoxide was eliminated at seal 4 on May 15, at seals 1, 2, and 3, on May 22. The analyses in both of these instances were succeeding analyses, so that although there is a difference of seven days, the elimination of the carbon monoxide may have occurred in one day.

This question, when is a fire out, is one we should know more about. From indications we believe that these fires were quickly extinguished after they were sealed; in other words, these areas could have been opened much sooner than they were; but the question is, how much sooner?

George S. Rice.—Has it been the experience in the anthracite mines that in a tightly sealed area, where there may be no fire, the oxygen content will disappear? That is the experience in the bituminous mines, as we have found repeatedly. The oxygen rapidly disappears, being either combined or absorbed by the carbonaceous material.

Douglas Bunting.—We have had no experience along that line. We have never made any analyses in closed areas free from fire, but my opinion is that the oxygen content will diminish; CH_4 would naturally increase.

J. B. Warriner,* Lansford, Pa. (written discussion†).—This paper deals with conditions in the Wyoming, or northern, anthracite field, where the veins are of moderate thickness and the pitch of the measures is light. Conditions in the southern anthracite field are markedly different. There the Mammoth vein attains a thickness of over 100 ft., under special conditions, and the average pitch of the measures is 70°. Also, a great deal of the workings are in gob areas, or areas previously mined and now being reworked. Most of our fires have occurred in such areas and in some cases the situation has been further aggravated by the fact that rolls or contortions of the measures have doubled the Mammoth vein on itself, forming what is known, in local parlance, as "capped anticlines."

* General Manager, Lehigh Coal Navigation Co.

† Read by Henry H. Otto.

Under these conditions, sealing or extinguishment of fires by smothering is practical only to a limited extent. Sometimes sealing is used in conjunction with other methods, but the method finally evolved is to fight the fire from above by means of water. It must be understood that these are not fires that break out into a blaze or that have the normal features of a conflagration; they are smouldering or creeping fires that advance slowly through the gob with limited air supply, and burst into a blaze only at points where the air supply is ample. Such fires are often difficult to detect until they have spread over large areas. In recent years, we have men go through the workings each night, after the miners are out, in an endeavor to locate any fires that may be starting.

The cause of many of the fires is obscure. Often these fires are not discovered until years after the workings in the areas have been completed. The most probable cause is the burning of powder in a loose hole, which is a fairly common occurrence in gob workings. The mysterious nature of these fires, however, has led to various ingenious theories as to their origin. There is still some leaning toward the theory of spontaneous combustion. It is believed also that gas accumulates in large open spaces in old workings, and is ignited by sparks from falling rocks, especially those containing iron pyrites.

The accepted method of fighting these fires is by putting water in at the outcrop or from an upper level, or by driving chutes in one of the small underlying veins and cross-cutting back to the big vein above the fire by means of rock holes or small tunnels. This latter system is necessary in any case in order to explore the fire area thoroughly and to be sure that the fire is entirely out. To do this requires the driving of a considerable yardage of chutes, headings, and tunnels. To illustrate the methods pursued in fighting these fires, a brief description of three will be given.

The Coaldale fire started in December, 1914. Here the Mammoth vein is on a 45° pitch. The fire was fought by direct methods, then by driving into the fire area at various points from the underlying Skidmore vein. Tunnels and chutes, the latter both in coal and rock, were driven to isolate and fight the fire. After the area of the fire was definitely determined, the gangways, tunnels, and chutes were sealed. The holes and crevices, as well as the space back of the dams, were filled with culm until the area was effectively sealed and the fire smothered. In addition to the pumps and ventilation expenses in connection with this fire, 1500 ft. of chutes and 2000 ft. of headings or holes were driven around the fire, as well as a tunnel to fight the fire, at an expense of $125,000. Thirty men were employed on each shift driving chutes and headings, six men carrying timber and supplies to the miners, in addition to an assistant foreman or fireboss to patrol the area and look after the safety of the men. In this case, it was possi-

ble to smother the fire by means of the seals on the gangway level supplemented by the culm poured in on top of the fire area with the water, which thoroughly filled all crevices and other openings, completely shutting out the air and also extinguishing any fire with which it came into direct contact.

This method of pouring culm mixed with water into a fire area has enabled us in the past 6 years to cool off and practically extinguish the

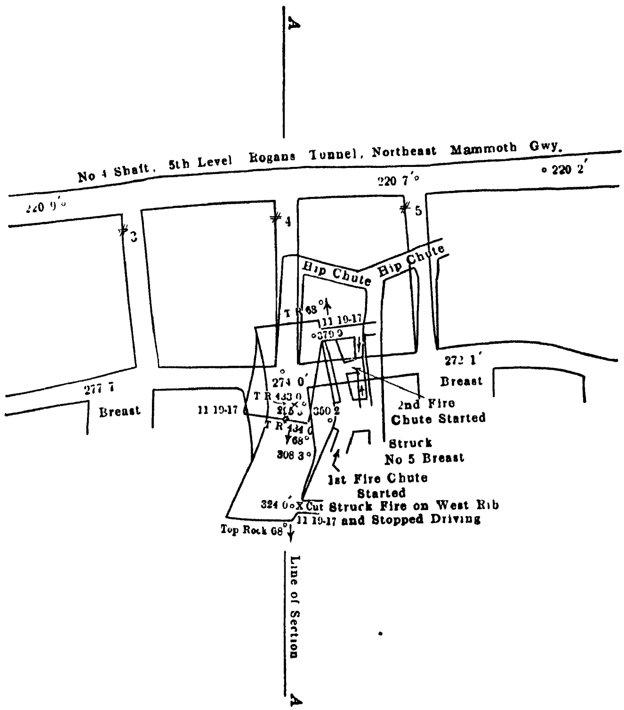

FIG. 5.—WORKINGS IN NORTHEAST MAMMOTH GANGWAY OFF ROGAN'S TUNNEL IN VICINITY OF FIRE DISCOVERED MAY 29, 1918.

entire area of the famous Summit Hill mine fire. In that area the fire is comparatively close to the surface and the vein is on a light pitch, so that the culm and water can be introduced into the fire area by churn-drill bore holes with little difficulty.

The Lansford fire, Figs. 5, 6, and 7, was discovered May 4, 1914, in the Mammoth vein, in the territory known as the Main Curve, No. 4 slope level. The vein in this area is in fault and caps. It is approximately 60 ft. thick and is on a 20° pitch. Rock holes and rock chutes were driven underneath the fire area, to locate it, and chutes were driven in the vein

itself. This was one of our first experiences with this kind of fire and it was decided that conditions were favorable for sealing the area.

It was thought impossible to shut off all air from the fire, because previous workings had disturbed and shattered the pillars, but it was thought advisable to shut off the air as far as possible. At one end of the fire area a complete seal could be made by the ordinary brick dam. At the other end, there was a large opening on the gangway level and various smaller openings some distance above. It was therefore necessary to build walls of considerable length in sections approximately 8 ft. high, each section being built upon the previous one, as soon as excavation into the loose coal could be made, so as to provide room. The total length of this dam, including the wing on the south end is 143 ft. The bottom

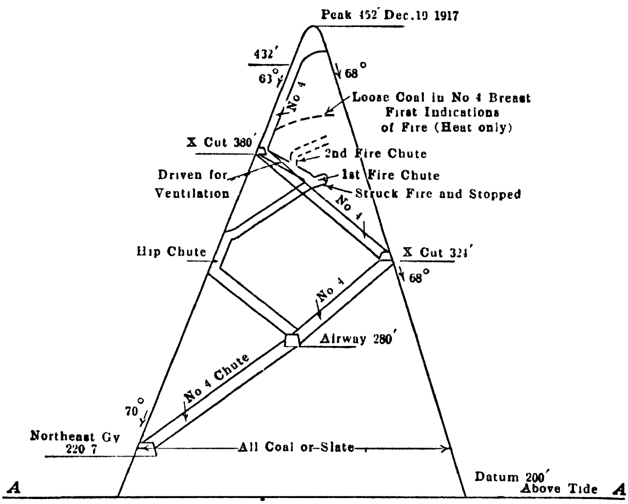

FIG. 6.—SECTION THROUGH No. 4 CHUTE AND BREAST, LOOKING EAST.

was laid about 6 ft. below rail elevation in the gangway. The height of the dam at the south end was 26 ft.; at the north end, 46 ft. The width of the dam at all places was 7 ft. The entire dam was reinforced with rail and iron rods. Five-foot hitches were cut into top and bottom rocks and 8-ft. hitches in the roof on the north end; see Fig. 8.

Excavation for the dam was made in solid coal, with the exception of the top and bottom rock hitches. Timbering was kept up to the face of excavation by means of 12-in. steel beams 11 ft. long and 8 by 12-in. timber 7 ft. long. The 12-in. steel beams were used as collars and 8 by 12-in. timber as legs. These sets were placed on top of each other as excavation progressed until the required height was reached. The legs were set perpendicular so that plank could be nailed inside and used for forms for the concrete. The 12-in. steel beams made good reinforcement.

Concreting was done in three sections, the work being started as soon as excavation would permit. The south section of the dam (50 ft.) was completed first; then the middle section (50 ft.), and finally the north section (43 ft.). Work on the dam was started three days after the fire was discovered, May 7, 1914, and was completed March 8, 1915. After the dam was completed 35 tons of silt were pumped through a 6-in. pipe in the dam, to put a good backing of silt against the wall. The maximum height of water in the dam was 49 ft. Five 10-in. bore holes were started from the surface but only two reached the vein, the others being abandoned by the contractor because of the almost insurmountable difficulties

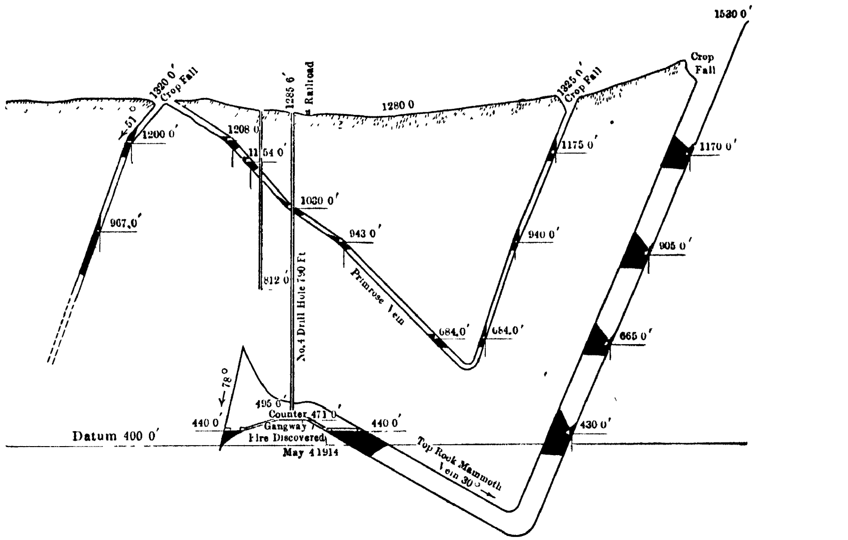

FIG. 7.—LOCATION OF FIRE IN ANTICLINAL OF MAMMOTH VEIN EAST SIDE OF NO.4 SLOPE LEVEL, LANSFORD COLLIERY.

encountered in sinking through 700 ft. of caved ground. About 1800 tons of silt were run down into the fire, and also back of the dams. Two years later, after the area was reopened, no traces of active fire were found. The cost of fighting this fire was $180,000.

The third fire really consisted of several fires, probably of separate origins, though of this we are not sure. Their location is in the Mammoth vein at Rahn colliery along the Sharp Mountain invert. The fire area is in what is locally known as the east "Q" gangway. Here the vein stands practically vertical and the coal is soft and friable. The vein at breast No. 160, where the first fire was discovered above the gangway, on Feb. 6, 1918, is 200 ft. thick, because of a roll. About a year later, a hot area was noted on the outcrop 600 ft. above the gangway. To fight

this fire effectively, it was necessary to drive several rock chutes from the Skidmore to the Mammoth vein before the best point to fight the fire was located. A fire tunnel was driven into the fire district from the Springdale workings, which are 400 ft. above those of the second level at Rahn, on which the east "Q" gangway is located. The crop coal was mined from this tunnel and water run down on to the fire from the Springdale workings. Water was also pumped to the outcrop and run down into the fire. The fire is in an area that has been partly robbed and extends down to the gangway. This condition prevents even the

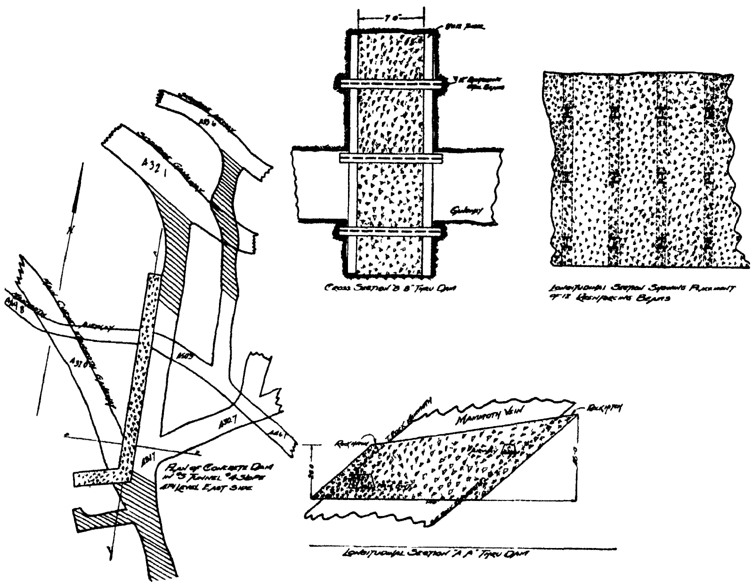

FIG. 8.—REENFORCED CONCRETE DAM IN NO. 4 SLOPE, 4TH LEVEL, EAST SIDE MAIN CURVE SECTION.

partial isolation of the fire by sealing. In September, 1900, a cave occurred at breast No. 143 on the gangway, and we have not been able to get in to the fire area since on the gangway level, though water is still being run on to it from the upper workings.

On Aug. 10, 1921, fire was discovered about 30 ft. up chute No. 33. It had evidently reached a point where sufficient air was available to cause a blaze. This alone led to its discovery, as no work had been done anywhere near the area for about two years. When this fire was discovered, steps were taken to keep it off of the gangway by means of clay stoppings, behind which water was pumped. This fire is 6600 ft. west of the one discovered in 1918 and 2,000 ft. east of the shaft. No

sign of fire has been noticed on the outcrop as yet. A 10-in. pipe line will be laid to the outcrop and water will be pumped into it. To locate the fire area definitely, and also to get above the fire, two rock chutes are being driven to the Skidmore vein from the Bottom Split gangway. One chute is 180 ft. east of the fire, and the other 140 ft. west of it. When the Skidmore vein is reached, two chutes will be driven from the rock chutes at an angle of 35°. These will intersect near breast No. 33 and up the breast about 150 ft. If a test chute from the junction point shows we are below the fire, we will then start a checker board system of chutes, until we get outside and above the fire. Water can be poured on it from the chutes. A pump has been placed in the ditch on the gangway, which

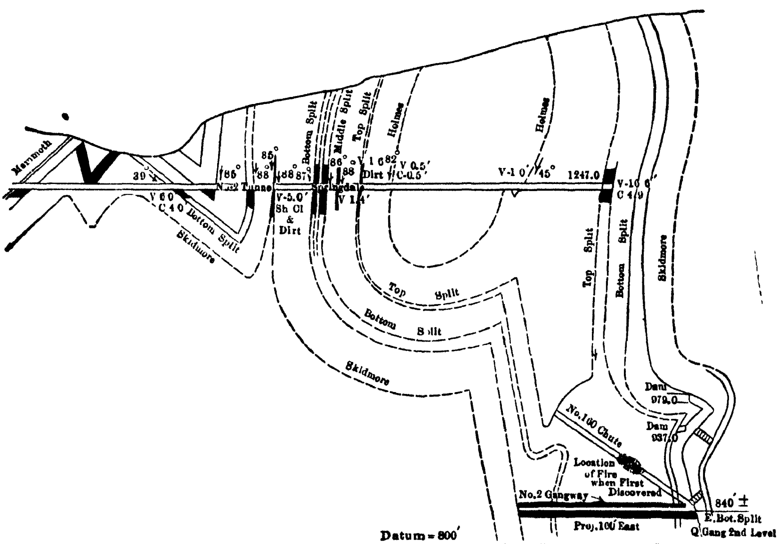

FIG. 9.—SECTION THROUGH BREAST NO. 160 OF MAMMOTH VEIN, EAST SIDE, SECOND LEVEL, RAHN COLLIERY.

will furnish all the water necessary to use on the fire, in addition to the water pumped in on the outcrop. Owing to the robbed condition of the workings, the fire cannot be sealed.

Several days after the fire was discovered in chute No. 33, a hot spot was noticed along the outcrop at a point 2000 ft. east of chute No. 33 Whether this is another fire or whether it is connected with either or both of the other two, we do not know. The cost of the fire at breast No. 160 to date has been $84,000. This does not include all the cost of special tunnels, etc., at Springdale, as coal was won from part of the fire area.

We also have occasional minor fires starting on the gangway, or elsewhere, but these, so far, have been extinguished with comparative ease.

Naturally, mine fires in heavy pitching seams are dangerous to the fighters on account of gas and the difficulties encountered in getting material to the point where it is to be used. Ventilation of the chutes and headings from which the fire is fought is also one of the greater difficulties. It can readily be seen that in any computation of the cost of mining coal, the cost of fighting fires is an important item.

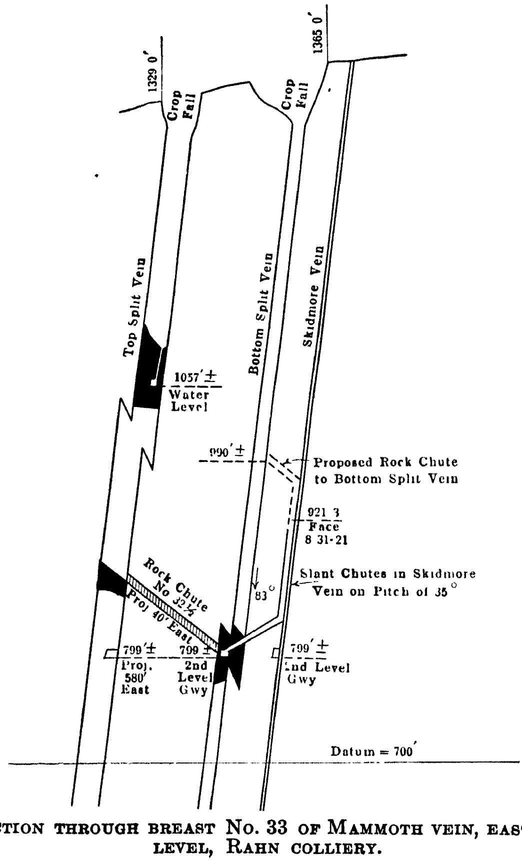

FIG. 10.—SECTION THROUGH BREAST NO. 33 OF MAMMOTH VEIN, EAST SIDE, SECOND LEVEL, RAHN COLLIERY.

DOUGLAS BUNTING.—Were any chemical analyses made of the air in these closed areas?

HENRY H. OTTO,* Lansford, Pa.—I do not think any analyses were made; I have only been at Lansford about a year and have not found any records of analyses in the reports in our files. Perhaps Mr. Ludlow can answer that.

* Mining Engineer, Lehigh Coal and Navigation Co.

Edwin Ludlow, New York, N. Y.—Such analyses were made.

Douglas Bunting.—Did they seal the return or the intake first?

Henry H. Otto.—I do not know. I have been there only a short time and these fires occurred about 1918.

E. M. Chance,* Philadelphia, Pa.—Up to the outbreak of the war, when I was more or less actively interested in fighting mine fires, I had made many hundreds of analyses of mine air in fire areas.

The presence of carbon monoxide *per se* indicates that there is fire. Of course, carbon monoxide may be found in mine air, due to the use of explosives. But, on the other hand, we may have fire in a sealed area, and particularly such fires as Mr. Warriner has described in the Lehigh region, where the combustion is slow, has extended over a long period, and where the combustion temperatures are low. Under these conditions, little or no carbon monoxide will be found, therefore, its absence from a sealed area is no indication of the absence of fire. If, on the other hand, we have a fresh fire, such as Mr. Bunting has described in the Wilkes-Barre district, where the combustion is brisk and the temperatures are high, carbon monoxide will almost invariably be encountered. This is due to the fact that carbon monoxide is not readily formed at low temperature. The reaction of oxygen plus carbon produces almost wholly carbon dioxide, at temperatures of 1400° F. or less, whereas with high temperatures, such as those that we have in brisk, active combustion in new fires, with ample air supply, there is a rapid formation of considerable quantities of carbon monoxide when the fire area is sealed and the air supply is cut off.

I believe a far more accurate criterion of the presence of fire in a sealed area is an observation of large quantities of carbon dioxide. If a section of a mine is sealed, the oxygen content will disappear almost immediately, irrespective of the presence of fire; the amount of oxygen will drop to less than 1 per cent.; in fact, it will disappear entirely if the stoppings are tight. That is characteristic of almost any sealed area, where the stoppings are tight, whether there is fire present or not; but if a fire is present, accompanying this drop in oxygen there will be a rapid rise in carbon dioxide. This is not true where there is no fire. The carbon dioxide content, under those conditions will be low, possibly around 2 or 3 per cent. These figures I am quoting from memory, but they are only a general indication of the difference between the analysis of a total black air and one caused by the absorption of the oxygen in the enclosed air.

When a fire is sealed and the stoppings are on and the pillars are in good shape, so that the air supply is cut off, combustion will be almost immediately stopped; but that does not mean that the fire is out. We

* Engineering Manager, Day & Zimmerman.

have two conditions in a fire—one is active combustion, the other is the presence of a mass of heated coal and heated rock. The combustion may be stopped immediately by the exclusion of fresh oxygen, but it takes time for the heat held by this mass of incandescent coal and rock to be dissipated by radiation; therefore, the gases characteristic of combustion may disappear. However, if the fire is of long standing, it will reappear, perhaps, when the workings are reopened. Therefore, it seems to me that the only safe guide in determining the extinction of a fire is to reopen the workings, remove whatever falls there may be, and examine them critically, because, although the combustion may have stopped, sufficient heat may be locked in the strata to renew combustion days or even months after ventilation has been re-established.

In sealing off a mine fire, extinguishing may be accomplished by creating a deficiency in oxygen. In the cases that Mr. Bunting has quoted, the workings were sealed off sufficiently tight for this aim to be realized, but this cannot always be done.

F. B. Nold,* Lansford, Pa.—The fire area in No. 4 shaft was sealed off from the other part of the mine; the intake airway was sealed first. The other fires described were entirely in old gob workings. The fires may have started a week or five months before they were discovered. Our experience with fires in old workings is that an analysis of the return air from that section of the mine does not show conclusively whether fire is present or absent. In the case of the fire in the east "Q" gangway of No. 11, samples of the return air were taken a number of times but the analysis showed no trace of carbon monoxide, although the fire was burning.

Edwin Ludlow.—The Summit Hill fire started in 1859 in the basin south of the Summit Hill anticlinal. That Summit Hill anticlinal dips to the west until it becomes a capped anticlinal, when it joins in the main basin, which is called the Lansford basin. The fire worked its way down toward this capped anticline. Various attempts were made to put out the fire. An open-cut was attempted, but the fire got around it. It was then decided to try to seal off the fire with drill holes and slush; this, however, was not successful for the fire worked its way around between the drill holes and to the west.

In 1908, a clay dam 12 ft. wide, with a cement core, was built across this basin and down to what was then water level in the mines (the coal was 250 ft. below the surface). The dam was carried down as an open-cut until there was over 50 ft. of surface, and was then carried in the vein itself by shafts 50 ft. apart, and the dam completed. This was considered entirely safe.

In 1912, not only was the fire burning up against this dam, but there

* Mining Superintendent, Lehigh Coal and Navigation Co.

was a good deal of steam to the west of it. After studying the outcrop of the fire and the type of the anticlinal, it was decided that it would be possible to strip off the surface, and recover the 50 ft. of coal lying under that anticlinal, which had been first mined a great many years before. This work was started as an open-cut. The temperatures, however, kept on increasing until at one time at 300 yd. west of this dam the temperature was over 550°. However, drill holes that were put down showed that the coal was perfectly cool. This temperature was in what might be called rock chimneys, or cracks in the rock over where the coal had been taken out and replaced by the dam, but the rock had been left above. The heat of the fire caused cracks and high temperatures in the openings in the overlying rocks, which had to be overcome in carrying on the stripping operations. We then started in to drive the fire back from the dam. Pumps were installed to take the breaker slush made at the Coaldale breaker, in the valley below, and were run during the 8 hr. that the breaker was in operation. It was found that this slush choked the drill hole, and it was necessary to follow that up with pumping clear water during the next 16 hr. By checkerboarding the fire area with drill holes, the slush was carried through the cracks in the rocks about 100 ft., making an air-tight seal that kept driving the fire back until it almost disappeared. The principal requirement was to keep the slush fine enough as it was put in, and to pump clear water during the night, in order to keep the slush clear and to fill the cracks in the rocks so far as possible. By following this plan, the fire was under control at the time I left Lansford.

H. M. Chance,* Philadelphia, Pa.—Nothing has been said in this discussion regarding fighting fires by flooding a mine. I have a story to tell which may be an old one to many members of the Institute, but one that we can well bear in mind when we have occasion to consider the possibility of extinguishing a fire by shutting off its air supply.

About 12 or 15 yr. ago, I visited the old Midlothian mine in the Richmond coal field. Its recent history was that there had been a fire in the old workings for two or more years, and mining had been abandoned, although the water had been kept out. The fire was on a 300-ft. level, I think, and the bed of coal had a pitch or dip of about 30 or 35°—possibly a little steeper in places. Attempts to shovel out the fire and to fight it with water had failed. The air-ways, entries, and gangways were small and did not provide facilities for fighting the fire by these methods. So it was decided to flood the mine. The workings were not large; possibly a length of about 300 to 500 ft. was on fire; this was above the 300-ft. level; there was no fire below that level. As the mine was comparatively dry and made water slowly, it took possibly two months for the water to

* Mining Engineer.

rise sufficiently to seal off the 300-ft. level. From that point on, the water rose rapidly in the shaft and quickly reached to within 18 or 20 ft. of the curb, so that the whole mine was thoroughly sealed by water. There were no outcrop holes, and the outcrop of the coal was below the natural water level of the country. The country is flat and there is a stream close by, running close to and above the place of the outcrop which is covered by soil and gravel thought to be about 15 or 20 ft. deep.

At the end of 18 months, it was thought the fire must certainly be out, so the water was pumped out, but the fire was still burning. Doubtless a large mass of rock and culm, probably including the formations above and below the coal bed, had become heated to a high temperature, and remained throughout the 18 months at a temperature at which active combustion recommenced as soon as air was admitted to the workings. The only solution seemed to be to flood the mine again, putting down a bore hole to allow the air in the old workings to escape, so that the water could reach the seat of the fire.

William L. Jacobus,* Wilkes-Barre, Pa. (written discussion).—Of the various ways of fighting mine fires, sealing is probably the one mostly used; this, at least, has been the experience of the Susquehanna Collieries Co. Our experience with the sealing of mine fires has shown that the methods of attack must be varied. In some cases we were able to seal off with stoppings; in others, we were only able to seal the bottom, or as it might be said, to shut off the drafts; in other cases, it was possible to do just the opposite, *i.e.*, shut off the top, or chimney. This last method is now being used at one of our operations. The lower part is open, but because of inaccessability, on account of old workings, and also in order to keep another level working, a booster fan is kept going; this holds up the air in the so-called sealed section.

Analyses of the air from the upper stoppings show that the oxygen is nearly exhausted and that the carbon monoxide is reduced to a minimum, showing that the upper stoppings are tight and that the fire is consuming the oxygen. Just as soon as this low-oxygen air reaches the fire, the fire will die out. Of course, as to when this actually happens is only guesswork as we have not seen the fire nor do we know its exact location.

When the fire was first suspected, and to get the approximate location and determine what airways to shut off, we tested numerous places with a carbon monoxide indicator; this showed where the stoppings should be placed. After the stoppings were put in, the air in the upper airways began to accumulate carbon dioxide and depreciate in oxygen. As the air from the fire accumulated, we obtained it farther down the pitch. Every two weeks, we analyzed the air back of the stoppings, and twice a week the foreman tested the air with the carbon monoxide

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* Chemist, Susquehanna Collieries Co.

indicator; this shows the increase, decrease, or disappearance of the carbon monoxide at various levels.

The reason for sealing is to keep fresh air from the fire; therefore the stoppings must be tight, whether they are placed to shut off the draft or the chimney, or to surround the fire. Concrete stoppings are excellent. Wood stoppings are good, but will dry out and leak even if cemented. For this reason we coat the outside with a plastic roof material made of asbestos fiber and asphalt. This will dry and always be gummy enough to accept any slight movement of the wood and still keep the stopping tight. This material can be applied to a stopping made of concrete, brick, stone, or expanded metal with a cemented coating.

It has often been stated that filling the fire area with some inert gas, such as carbon dioxide, nitrogen, etc., would extinguish the fire. This is true, if the section is sealed tight; but as the area to be filled with gas is large this method would be costly. As the products of combustion are the worst enemy of the fire why not seal tight and allow the fire to put itself out? If the fire is large, the results will be obtained more quickly. If the fire is small and confined in a large sealed area, the time necessary to consume all the oxygen will be so great that it will pay to give the fire air so that it will increase in size in order that when the air is shut off the time required to consume the oxygen will be reduced considerably. This has been done several times to our knowledge.

A fire located in an old abandoned section, where the strata are broken through to other old veins or to the surface, is one of the most difficult to cope with. Under these conditions the best plan is to determine and seal the inlets and then try to put air depleted of oxygen, such as stack gases, into the main intake.

When sealing a mine fire, while the last few stoppings are being placed, mine gas may accumulate above the fire and explode, blowing out these stoppings. These explosions may occur at regular intervals or the intervals may differ in length. Quickly replacing a blown-out battery will lengthen the interval between explosions, but at one of our fires the batteries were so located that it was necessary, after quickly repairing them, to get so far away one could not tell just when the explosion occurred. It was then necessary to crawl back cautiously to investigate; if the battery was wrecked there was no way of telling how long before the explosion had occurred. To overcome this uncertainty and to know the exact time of the explosion, so that we could immediately return and rebuild the stopping, the inside foreman built a wooden frame on all sides of the opening outside of the wrecked stopping and fastened a small roll of canvas at the top of this frame. When the explosion occurred, the vibration unrolled the canvas, forming a canvas trap door, thus preventing a large amount of air from immediately rushing in to feed the next explosion. The falling of the curtain also closed an elec-

trical connection causing a bell at the foot of the shaft to ring. After a few explosions the intervals became greater, until we were able to build the permanent battery.

J. A. S. Ritson,* London, Eng.—We have two kinds of mine fires in Great Britain, one caused by the use of naked lights, and the other, which is much more prevalent and serious, caused by spontaneous combustion. The spontaneous combustion fires occur in both the open-light and the safety-lamp pits. We have no definite rules for the treating of mine fires—in each case the treatment depends on the conditions but generally sealing off is adopted. Until quite recently, we attempted to dig them out, but have decided that sealing off is the best method.

Fires caused by spontaneous combustion in the safety-lamp pits are the most serious and the most common. In the deep mines of South Yorkshire (which vary from 700 to 1100 yd. deep) there is a fire in some form or other all the time. We try to catch a fire at the first sign of heating before any flame or any red heat has developed. We have tried various ways of detecting when it begins to heat, such as taking samples of the return air and having them analyzed, etc., but we have returned to depending on the nose. The smell of gob stink, to those who know it, can be detected with the nose much more quickly than by any other method. As soon as gob stink is detected, the men from that district are withdrawn, and that area is thickly coated with stone dust. You must remember that these pits are hot, deep, dusty, and very gassy; they are all worked longwall. The first thing we do is to dust thoroughly all that area, right to the face, with stone dust; then build off the affected area. The building off is done by coating all the longwall buildings, called pack walls, with loose sand. If this is put on by hand and pushed into all the crevices, it makes a comparatively good air-tight stopping. At the same time all the roads into the particular area (which is generally about 100 yd. square) are built off. We put in a stopping of loose dirt first, then a building of sand, then a brick wall, then outside of everything we put in sand, piling sand all around until the air is wholly excluded. While building off, we restrict the air reaching that area; we do not cut it off wholly, we regulate it. The last thing which is done is to build the return side. The building off of the last stopping is generally done with breathing apparatus either self-contained or helmet type. Of course, we are provided with birds.

In dealing with spontaneous fires in non-gassy mines, we use much the same precautions but do not withdraw the men because there is little gas made, and most non-gassy mines are also damp. Sometimes, a fire occurs in the middle of a pillar, in which case we dig it out. In digging out fires, we invariably attack them on the intake side taking a plentiful

* Inspector of Mines of Great Britain.

supply of air and water with us. Brattice is taken up and the men work on the fresh side, the fumes going down the other side. If necessary, breathing apparatus is used.

As regards spontaneous fires that have been sealed, we have no data as to whether or not those fires are out. Areas in the east of Scotland that have been opened after 40 yr. have blazed again within a very short time. These are seams near the surface; so it has been argued that the fire draws air through cracks, from the surface, and that it is practically impossible to get them out. We are working places like that. When we reopen them, we open up only a part at a time and as we advance we fill in the waste with flushed material from the surface—flushed sand through pipes. The water is pumped back to the surface again and the small stone, sand, and other stuff, is left. That makes a very tight joint. Many fires occur in open-light pits from the carrying of open lamps in the cap. The method we are trying to adopt in these mines is the withdrawal of those lights and the substitution of some sort of electric lamp. In the case of a fire caused by a naked light, we almost invariably (unless it is impossible) attack it from the intake side with water and dig it out. If that fails, we seal it off by dirt or brick stoppings, always with sand on the outside.

R. V. Norris, Wilkes-Barre, Pa.—At the National colliery, Taylor, Pa., a mine fire was extinguished by excavating around it a trench about 50 ft. wide at the bottom and about 150 ft. at the surface.

R. D. Hall, New York, N. Y.—I was interested in Mr. Ritson's description of the methods by which they fight fires in England, but cannot quite understand how, as they advance, they fill in behind them. I would very much like to know just exactly how they perform the filling-in which they do with slush of some kind.

J. A. S. Ritson.—What I meant was; we go into the area, which very often is longwall, and leave two roads in through the waste, which is composed of flushed material from the surface. By doing that, in case of fire, all that we must do is to block with brick and sand those two roads that we have left through the flushed material. This flushed material sets tight within 24 hr. We put up small, light, canvas screens with wood backing to them, and fill the flushed material behind them; within 24 hr. the wall is set tight enough for the screens to be taken down.

George S. Rice.—The fire fighting carried on at Anaconda mine in Butte, Mont., is a remarkable piece of work. As many of you may know, there is a great mineralized zone at Butte which years ago got on fire and gave untold trouble. The high temperatures and the fires were most extensive and shut off a most valuable part of the mine.

A number of years ago, it was decided to fill in with slimes, and that work is being conducted on a very large scale. A large force of men is continuously at work, using a half dozen diamond drills and three cement guns filling the old stopes, from the bottom up. Many of the stopes have already been cooled off and valuable pillars recovered; but the particular points of interest are the combination of this filling through diamond-drill holes, forcing the slimes in under pressure, and the extensive use of the cement guns. Owing to the wide area of the affected zone, there is a movement of the ground and the rocks were all cracked, so that ordinary sealing and plastering was not effective. The cement gun proved an admirable means of filling in the cracks, and in some places where the ground was too broken, brattice cloth supported by chicken wire has been placed along the wall as a backing for the coating put on by the cement gun. The thin cement wall thus built up stands out from the rock wall and has a certain amount of flexibility. It is carried back a long distance from the stopping to prevent air leaking through broken ground. From observations made at different times during the last few years, I can say that this is a most admirably conducted plan and is well worth the attention of the colliery engineers, who have had so much trouble and difficulty in controlling fires in thick, pitching beds.

Mechanical Mining of Anthracite

By Herbert D. Kynor,* Scranton, Pa.

(Wilkes-Barre Meeting, September, 1921)

By the term mechanical mining is meant that operation, or series of operations, that replace the hand methods of mining. The first undercutting machine to operate in anthracite was placed in the Butler mine of the Hillside Coal and Iron Co., Dec. 23, 1910, by the Sullivan Machinery Co. The first Jackhamer used in the anthracite field was installed at the Wadesville colliery of the Philadelphia & Reading Coal and Iron Co. in Oct., 1912, by the Ingersoll-Rand Co. The first longwall conveyor was installed at the Dodge mines of the Delaware, Lackawanna, & Western Coal Co. in March, 1912. The first mechanical scraper equipment was installed in the Seneca mines of the Lehigh Valley Coal Co. in January, 1914.

The amount of anthracite mechanically mined since the first mining machine was installed in 1910[a] is as follows:

Year	Penna. Coal Co. and Hillside Coal and Iron, Tons	D L. & W , Tons	Scranton Coal Co , Tons	Temple Coal Co , Tons	Hudson Coal Co , Tons	Total, Tons
1920	372,737[b]	367,000[d]	322,892	74,127	371,878	1,508,634
1919	950,786	no record	306,616	81,575	319,356	1,658,333
1918	917,778	744,966	24,880	44,756	288,781	2,021,161
1917	757,940	838,800	5,050	57,279	164,180	1,823,249
1916	286,780	860,663	13,960	114,173	101,475	1,377,051
1915	428,819	592,169			8,253	1,029,241
1914	219,188	153,896				373,084
1913	171,505					171,505
1912	124,267					124,267
1911	40,319					40,319
1910	1,184					1,184

[a] A slightly greater production was obtained than here indicated, as the tonnage from individual operations is not shown.

[b] Decrease due to strike conditions in 1920.

[c] Does not include tonnage of D., L., & W.

[d] D., L., & W. decrease in 1920 due to having fewer mining machines in operation.

There are about 150 undercutters in use in the anthracite region, 80 mechanical loaders, and 10 longwall conveyors.

* Assistant to the Vice-President and General Manager, The Hudson Coal Co.

Coal Cutters

Shortwall Machine

This is a continuous cutting machine and is used in chamber work. The machine makes one clean cut across the face without withdrawing the cutter bar; the props may be set within 5 to 6 ft. (1.5 to 1.8 m.) of the face. There are four makes of this machine: The Jeffrey, Goodman, Sullivan, and Morgan-Gardner. These machines are similar in deigsn and principle, but differ in detail. The motors are rated from 30 to 50 hp. for 1 hr. continuously. We have found that cutter arms 6½ ft. long suit our conditions best. These arms are narrow, reducing the possibility of being caught under the coal and allowing more freedom for guiding the arm upwards or downwards in cutting.

The details of the chain design differ somewhat in the different makes of machines. The chains have blocks and straps so pitched that they engage with the sprocket wheel on the arm. The bits are held in the blocks by setscrews. The bits are either pick or chisel pointed, or a combination of the two, and are arranged in three to nine positions to suit the cutting conditions. The best satisfaction in anthracite mining is obtained with a pick-point bit and preferably located in all nine positions.

Longwall Machine

This machine is especially adapted to longwall mining, although, if the height is good and the roof sound, a shortwall machine may be used for this work. The particular advantages of this machine are: A thinner vein can be worked, as the height of the machine is only 18 in. compared to a minimum height of 24 in. on a shortwall machine. Also, propping can be done closer to the face with a longwall than with a shortwall machine, as the cutter arm can be held at right angles to the body of the machine by a locking pin. The cutter arm is reversible, that is, it may be turned about 210° so as to cut either right- or left-handed. The cutter arm may also be locked in a central position for transportation from place to place. Another advantage is that only one rope and one jack are necessary, as the body of the machine will hug the face of the coal. A shortwall machine requires two ropes and two jacks.

Drills

Jackhamer

The Jackhamer is an air-driven machine that may be carried by the miner from place to place. It is 18 in. (45 cm.) long with a cylinder diameter of 2½ in., a stroke of 2 in., an air connection for ¾-in. hose, and weighs 40 lb. Newer types, which weigh only 20 lb., are on the market. The machine is simple in construction and the maintenance charges are

low. A rifled bar and ratchet located in the back of the cylinder impart rotation to the piston, which in turn imparts rotation to the drill. The spring device on the lower end holds the drill in the machine. Steel drills up to 12 ft. in length may be used. Twisted drills are generally employed for drilling coal and solid hexagonal chisel-pointed bits are used in rock.

Electric Drill

There are two distinct types of electric drills, namely, a direct-driven electric drill of the auger type and a drill where the electrical mechanism drives a small air compressor, which in turn runs the drill as a percussion drill. An effort is being made to secure a type of electric drill that will replace the Jackhamer, particularly on account of the expense of compressing and piping the air over great distances; electric drills may be connected to the trolley wire at the foot of the chamber.

It is essential, however, that this type of electric drill have the following properties of the Jackhamer: (1) Its weight should not exceed 50 lb. This is particularly essential in low-vein mining as it is often necessary for the men to travel to the working faces from the gangway on their hands and knees, and to transport any object of greater weight than 50 lb. would be cumbersone and awkward and place a physical strain on the men. (2) It should drill in rock as satisfactorily as it does in coal. (3) It should be adapted to work under both wet and dry conditions. Of the electric drills that have been experimented with in the anthracite region, three have the required portability but do not drill in rock as satisfactorily as in coal. These types are manufactured by the Chicago Pneumatic Tool. Co., Pneumelectric Machine Co., and the Howell Drill Co. All are of the direct-driven auger type.

Experiment seems to indicate that the light-weight auger-type electric drill will not touch rock without a torque sufficient to throw the drill out of the operator's control. This electric drill does not work as satisfactorily when drilling holes under wet conditions as it should, due to short circuiting and shocking the operator.

Some heavier types of auger drills work satisfactorily in both coal and rock, but they are not adapted to thin-vein mining on account of their weight and the number of men required to handle them. The same objections apply to the percussion drill, which is heavy and cumbersome.

Mechanical Loaders

Scraper Loader Engine

Various types of hoists have been used for this work, the principal ones being the Vulcan, Lidgerwood, and Pneumelectric. The variation in horsepower of the motor ranges from 7½ to 25 hp. The essentially different types of hoists used are the portable and stationary.

The portable hoist is mounted on a truck, which runs on the mine tracks and can be transported along the gangway from one chamber to another. The advantage of this hoist is that, as it can be placed directly at the foot of each chamber, there is less friction on the ropes than there would be if it were set in a crosscut, necessitating fleeting of the ropes along the gangway. It requires an additional track in front of each chamber, however, so that the mine cars can pass in front of it. This necessitates widening the gangway at these points, which is a disadvantage, as the roof often will not permit of a wider gangway than the average 12 ft. The cost of lifting the necessary rock adds to the expense of the operation.

The stationary type of hoist has proved most satisfactory and has the smallest maintenance charges. This hoist is set in a crosscut between the gangway and airway, and is then able to handle the scraper in at least five chambers. It consists of two drums of equal dimensions and interchangeable. One drum carries the main rope and the other the tail rope; or in other words, one drum winds on and the other reels off the cable. The process is reversed depending on whether the scraper is going up or coming down the chamber. The drum-shaft gear is made in halves, providing for easy replacement. The engagement and release of the drum are made with one continuous movement of the engaging mechanism, thus making possible instant reversal of the scraper at the point of dumping into the car.

Scraper Equipment

The scoop is a V-shaped bottomless appliance that drags the coal along the bottom from the working face to the car and the gangway. It is built of steel plates, the lowest plate on the sides being slightly curved inwards so as to scoop the coal. The sides are braced near the open ends with a piece of channel or bar iron. The so-called medium scoop, which is in general use, is 4 ft. (1.2 m.) long, 4 ft. wide at the open end, and 17 in. (43 cm.) high, and has a capacity of 1000 lb. (453.6 kg.) of coal.

The jack screws are made up of 2-in. (5-cm.) pipe and are adjustable, being equipped with a screw, using nut and collar. These screws are placed at the working face and the snatch block pulleys through which the rope is drawn are attached to them.

Two types of pulleys are used: The snatch-block pulley is used for the face and the prop pulley for carrying the rope along the gangway to the foot of each chamber.

Conveyors

The form of conveyor most used for anthracite is the chain-and-trough type, which is used altogether in longwall mining. The links are from 9 to 12 in. wide and move in a steel-plate trough with flaring

sides. The return run of the chain is made over an angle-steel track located under the trough. The trough is made in sections from 12 to 16 ft. long supported on steel legs, or stands, and the conveyor is operated by an electric motor placed near the discharge end. The speed of travel is from 90 to 125 ft. per min. The full length of the conveyor may be shifted with bars so that it is kept close to the working face at all times.

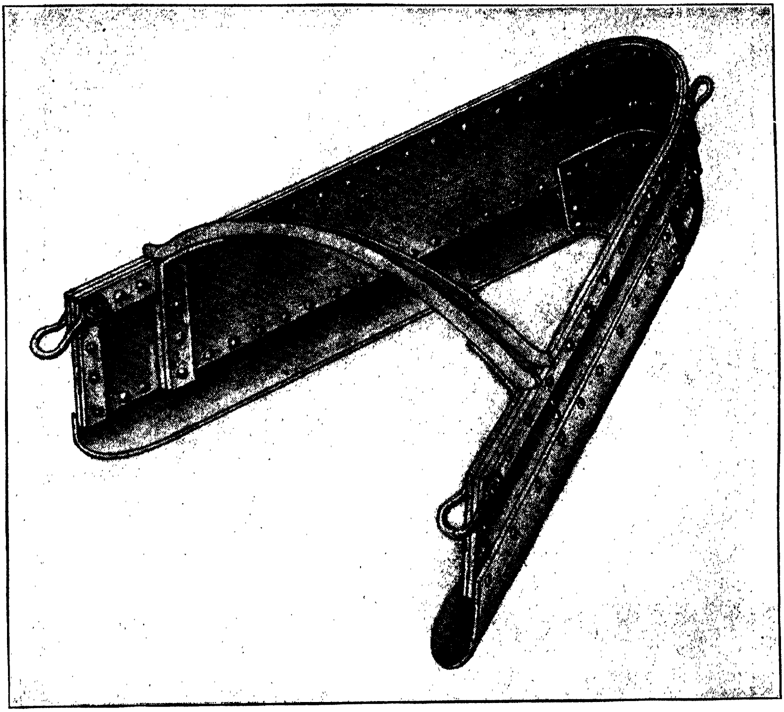

Fig. 1.—Perspective view of scraper.

Bumping Chutes

The essential details of operation of the bumping chutes are that one motion of the motor pulls the trough up the pitch for a few inches and the reciprocating motion releases it. This reciprocating motion is artificially created by the use of wheels in an incline trough. The conveyor thus abruptly falls away after the material has been carried forwards and so shifts its contents along the trough. The operation is repeated on the next cycle of the motor.

Shoveling Machines

The use of shoveling machines is practically the application of steam-shovel methods to underground work. The usual type of steam shovel,

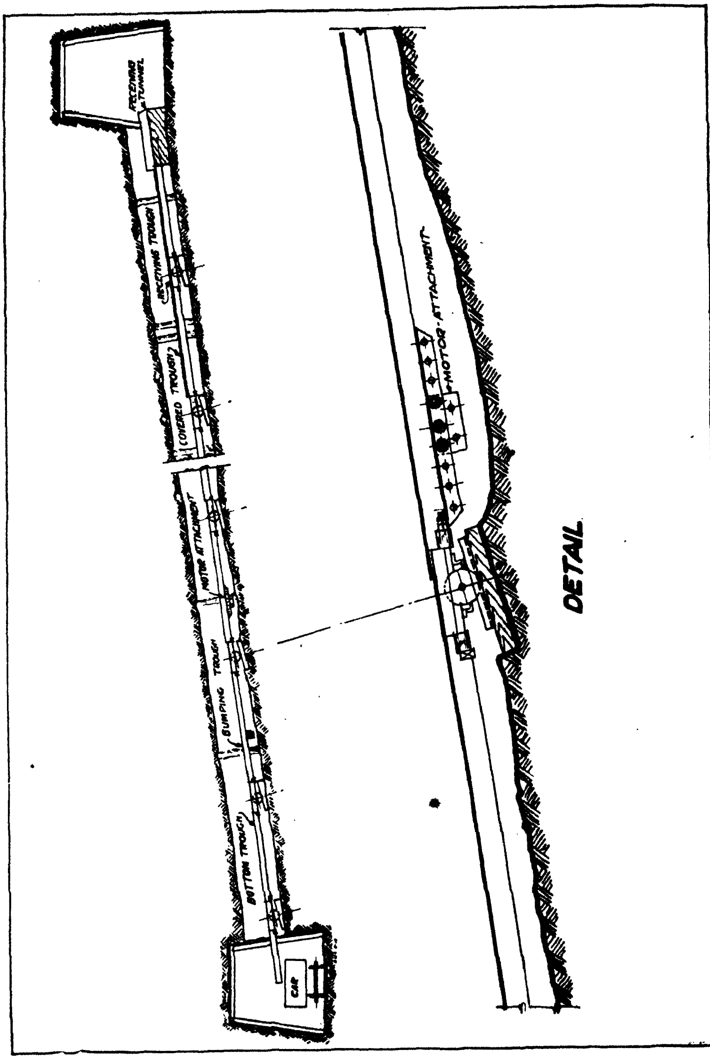

FIG. 2.—BUMPING TROUGH CONVEYOR FOR INCLINED SEAMS.

however, has been modified in general arrangement and design so as to be capable for operation in underground chambers. The shovel is electrically driven, provided with a boom and bucket, and moves on a

caterpillar truck. These machines, however, can only be utilized in thick beds, their design precluding good performance in beds less than 6 ft. thick. Where there is a large tonnage to load this machine can do considerable work; it has been known to load at the rate of 20 to 40 tons

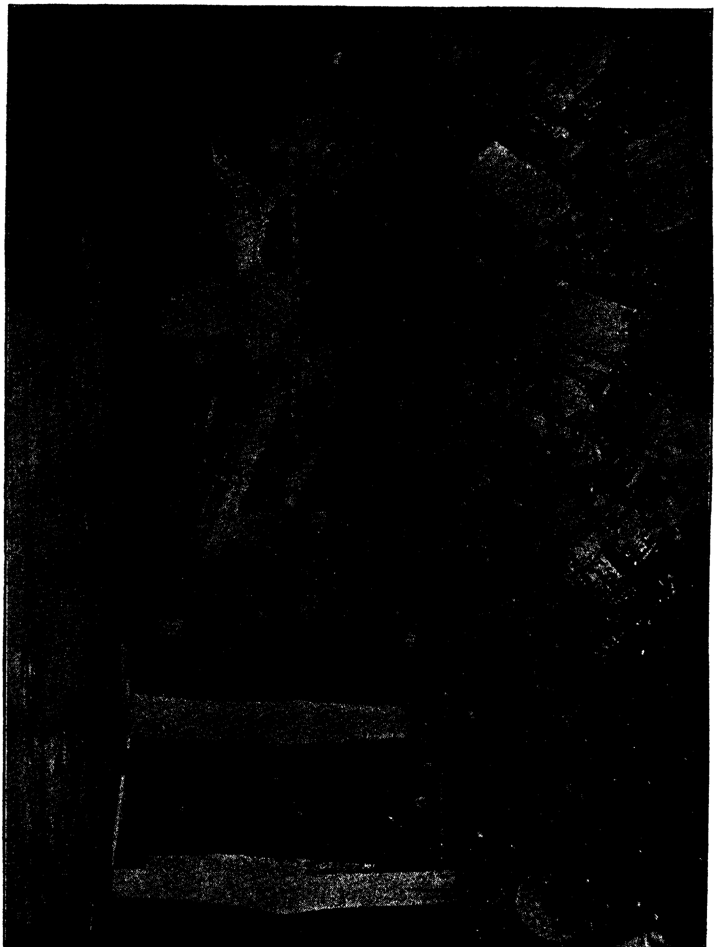

FIG. 3.—MYERS-WHALEY SHOVELING MACHINE IN A 4½-FT. SEAM.

per hour. The success of any mechanical loading machine depends on continuity of operation. When there are only a few tons per place and much loss of time traveling from room to room, the efficiency is low. Further, they are bulky and awkward to handle in any but the highest and best of mine roads.

The Shuveloder is a compressed-air operated machine, comprising three main elements: a truck with wheels suitable to the gage of the track; a platform, or turntable, which provides lateral movement to the shovel; and a body member containing the operating cylinders and shovel guide members in which the shovel and its arms function. The body piece consists of four cast-iron cylinders and their pistons, three operating valves, and a crosshead that travels in horizontal guides and carries the rope sheaves to which are fulcrumed the dipper arms and the dipper.

In action, the dipper duplicates the operations of the man shoveling by hand. It forces itself into the pile of broken material, alternating the

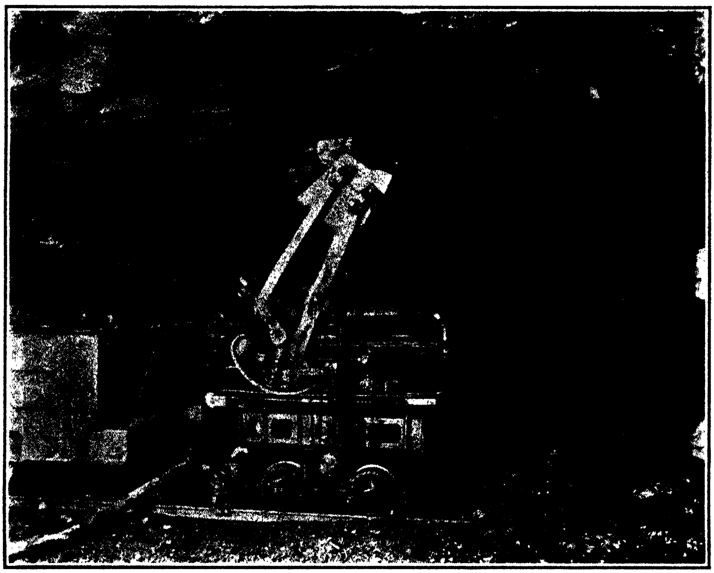

FIG. 4.—SHUVELODER IN OPERATION.

forward push with a lifting action. The machine is controlled by three hand levers located on the side of the machine. Air is admitted successively into the bottom cylinder, two center cylinders (acting as one), and the top cylinder. The bottom cylinder pushes the body piece forwards crowding the dipper into the pile (this is called the crowding stroke), the two center cylinders turn the rope sheaves bringing dipper up through the pile to a horizontal position (this is called the digging stroke), the top cylinder pulls the crosshead back, causing rope sheaves to turn, carrying the dipper up and over, where the load is discharged into a car at the rear of the shovel. The material is at all times discharged into the center of the car, since the two concentric tracks on which the body swings have a center in common with the center of the car. Five-foot sections of track

are used in order to move the machine up to the pile; when six sections of track have been used, a regular 30-ft. (9 m.) rail can be laid so that the empty cars can be supplied to the machine.

The machine has been designed for hand propulsion only; the short wheel base and light weight make it easily moved by the loading crew. The machine may be operated in a heading as small as 5 ft. (1.5 m.) wide by 7 ft. (2.1 m.) high or, when desired, a maximum lateral reach of 5½ ft. on each side of the center line of track can be obtained. The loading crew consists of two men, one man to operate the machine and one man to keep the empty car up to the machine.

Methods of Operation

Driving Chambers

If the coal is to be loaded into the cars by hand, the vein must be at least 4 ft. thick as the track is laid in the chambers, and to take up

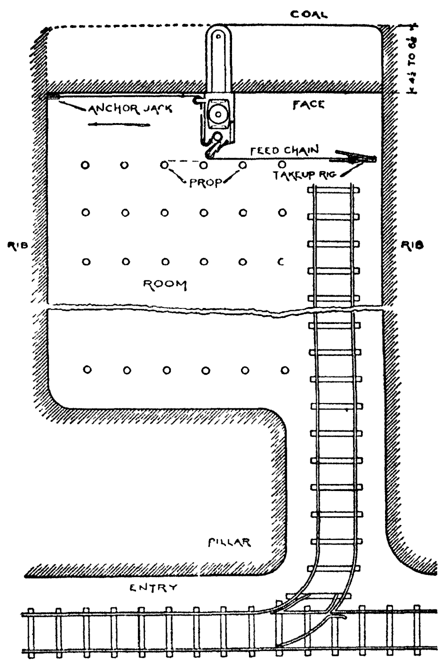

FIG. 5.—PLAN OF CHAMBER WITH SHORTWALL MACHINE CUTTING ACROSS FACE.

much bottom rock for the height would delay the operation. The conditions that tend to make the use of the mining machine a success are: good power, a practically level bed, a good roof, very smooth bottom,

conditions such that wide chambers may be driven. This latter depends on the depth of the bed under the surface, the nature of the roof, and the nature of the overlying and underlying beds.

The cycle of operation in undercutting in chambers by this method is as follows: (1) Unloading and moving machines to the face, (2) sumping the cutter arm under the coal, (3) undercutting across the face, (4) moving out and loading the machine on to the truck, (5) traveling to the next chamber. The machine moves from working place to working place on a self-propelled truck actuated by a direct transmission from the motor of the machine itself to the track wheels of the truck.

After the undercut has been completed the Jackhamer men drill the holes. As a general rule there are four holes in a 30-ft. (9 m.) chamber.

FIG. 6.—MACHINE IN LEFT-HAND CORNER OF CHAMBER.

To drill one 6-ft. hole requires 4 min.; to drill the four holes and move the drills to the next chamber requires about 40 min. After the holes are drilled, the miner and his helper clean out the drill holes and place the charge, tamping, and wiring. The holes are fired by an electric battery. The organization required for this method of working consists of a machine runner and helper and a machine miner and helper. The number of hand loaders depends on the number of chambers being worked and the number of machines in operation. As a general rule, one mining machine will keep sufficient coal cut for six hand loaders.

Where mining machines are used in conjunction with mechanical scrapers, machines are taken from face to face by their own power through the crosscuts, thus eliminating the use of a truck.

Coal Blown off Solid, using Jackhamer and Mechanical Loader

It has been found an advantage in many places, where conditions are not favorable for electric coal cutters and where the air supply is good, to use Jackhamers to blow the coal off the solid and load by means of a scraper. The miner drills the whole face of the chamber, as a rule taking from ten to sixteen 6- to 8-ft. holes, depending on the nature of the coal and the cleavage. The greatest difficulty encountered in this work in thin veins is the firing, where squibs or instantaneous electric exploders are used. As it is necessary to fire every hole or every two holes separately, a great deal of time is consumed both in drilling the coal and in

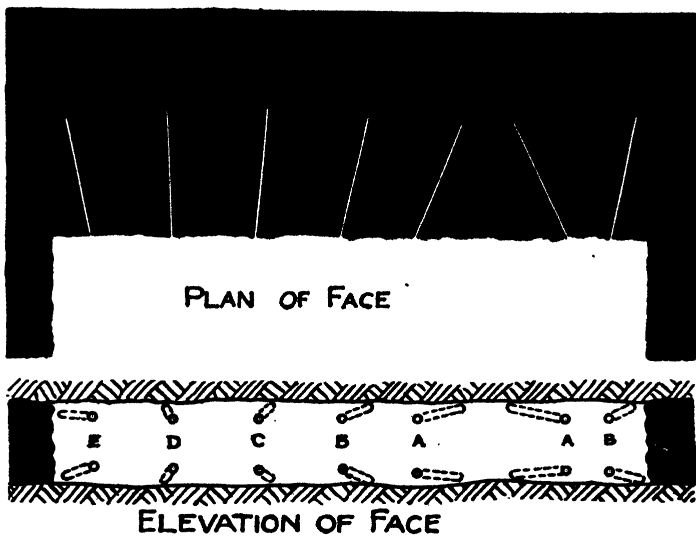

FIG. 7.—ARRANGEMENT OF DRILL HOLES WHEN USING JACKHAMER AND MECHANICAL LOADING.

waiting for the smoke to clear up so that the miner can return to the face. The following method was found to overcome this difficulty very satisfactorily:

The face was drilled with two or four opening, or cut holes, *A*, Fig. 7, and then the rest of the holes *B*, *C*, *D*, and *E* were drilled as relief holes. Holes *A* were fired with instantaneous electric exploders, and then holes *B*, *C*, *D*, and *E* were charged, connected, and fired by one pull of the battery, using electric delay-action exploders, each delay being 6 sec. The exploder placed in holes *B* go off first, followed at 6-sec. intervals by those placed in *C*, *D*, and *E*, respectively. By means of these exploders, two sets of firing will blast an ordinary chamber face.

The miner goes back into the smoke only once and the firing time is reduced 70 per cent.

In Jackhamer mining for scraper loading, the opening hole should be placed in line with the scraper roadway, or chute way. As these holes are fired first, the relieving holes will throw the coal toward this opening,

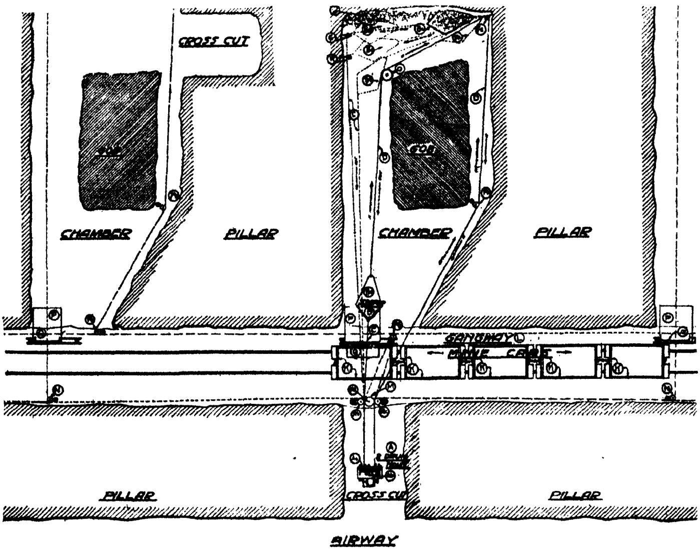

FIG. 8.—GENERAL LAYOUT OF TWO-DRUM HOIST AND SCRAPER LINE.

thus making the scraper work easier; it will also tend to keep the coal out of the gob. Gathering and loading coal by means of the scraper is probably one of the most practical and simple methods used. The scraper loads itself and does not require hard labor on the part of the operators.

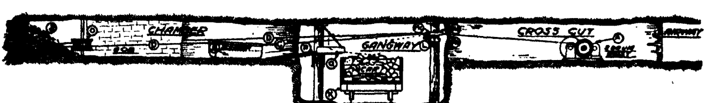

FIG. 9.—SECTION OF TWO-DRUM HOIST THROUGH CHAMBER AND GANGWAY.

As a rule five adjacent chambers that turn off the gangway in the same direction are worked as a unit. The hoist is placed in a crosscut opposite the middle chamber, if possible, and on the opposite side of the gangway. The standard, erected in front of the hoist and equipped with pulleys, serves to turn the ropes along the gangway to the foot of whichever chamber is to be worked. The load platform, one prop pulley for the main rope, and one for the tail rope are located at the foot

of each chamber. If the chamber is on the same level as the gangway, some top rock will have to be blown down to accommodate the loading platform, which consists of two props spaced about 4½ ft. (1.3 m.) apart with planking leading up from the floor of the chamber to about 1 ft. above the top of the car. A runway, or scoopway, is built along one side of the chamber; this consists of planks fastened to a row of props placed about 10 ft. from the rib and terminates near the face with a wooden drum pulley around which the scoop is drawn on its way to the pulley in the uppermost corner of the chamber. This pulley is held in place by an adjustable jack screw. An additional pulley and jack screw are placed at the opposite rib at the lower end of the loose coal. In addition, where the propping is close to the face, and in order to turn the

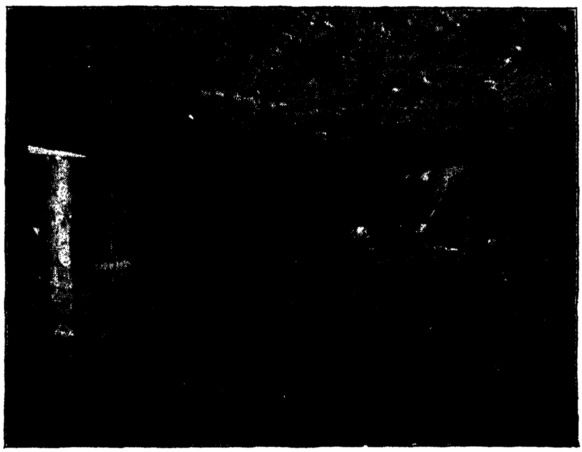

FIG. 10.—SCRAPER STARTING FROM FACE OF CHAMBER.

scoop without upsetting, a third jack screw is located along the scoopway rib several feet nearer the loading platform.

The tail rope is taken through the various sheaves up to the face and attached to the closed end of the scoop, which has been placed on the loading platform. The main rope is attached to the chain at the front, or open end, of the scoop. An electric-bell system is used for signalling. A push button at the face is used to regulate the movement of the scoop and one at the loading platform is used to stop the scoop and return it. The scraper system is usually operated by four or five men. The engineer stops and starts the hoist; the car man spots the cars in front of the loaded platform and signals the engineer as the scoop deposits its load into the car. Two, and sometimes three, men are needed at the face, depending on the amount of gob to be moved or the amount of timbering to be done; two men are all that are required for the actual operation of the scoop.

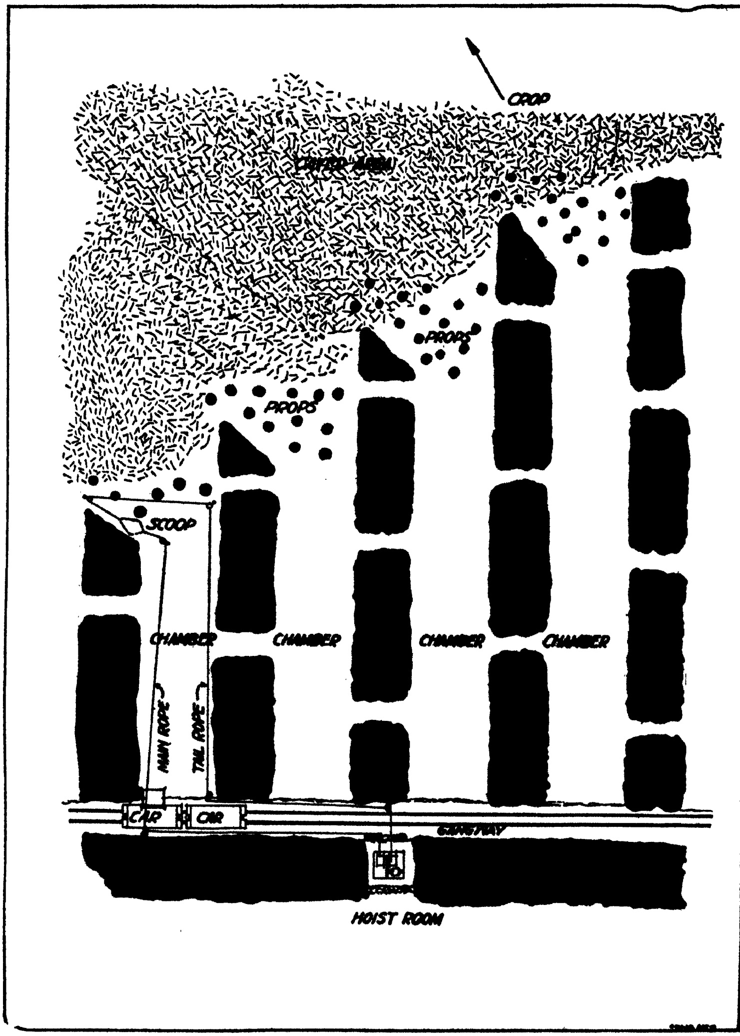

FIG. 11.—ROBBING WITH SCRAPER.

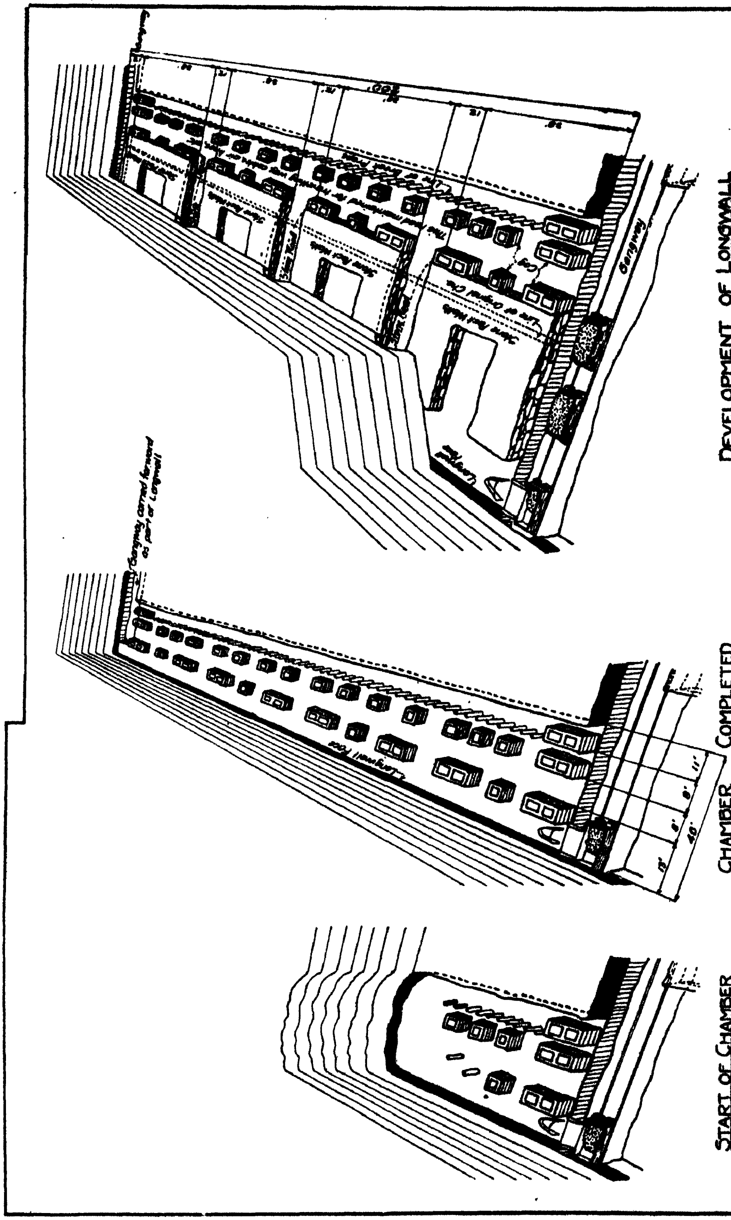

FIG. 12(a).—LONGWALL WITH ROCK-PACKED ROADS.

As the scoop approaches the uppermost sheave in one corner, a signal is given to stop. One of the men then places the main rope on the sheave in the opposite corner and signals to pull the scoop across the face. When the scoop is loaded, a stop signal is given. The main rope is taken off the sheave, which is an open snatch block, and a signal is given for the scoop to be pulled to the car.

The advantages of the scraper loader are: (1) It eliminates taking up rock in chambers, and the cost of transportation and storage of this rock whether inside or out. (2) It saves trackwork in the chambers.

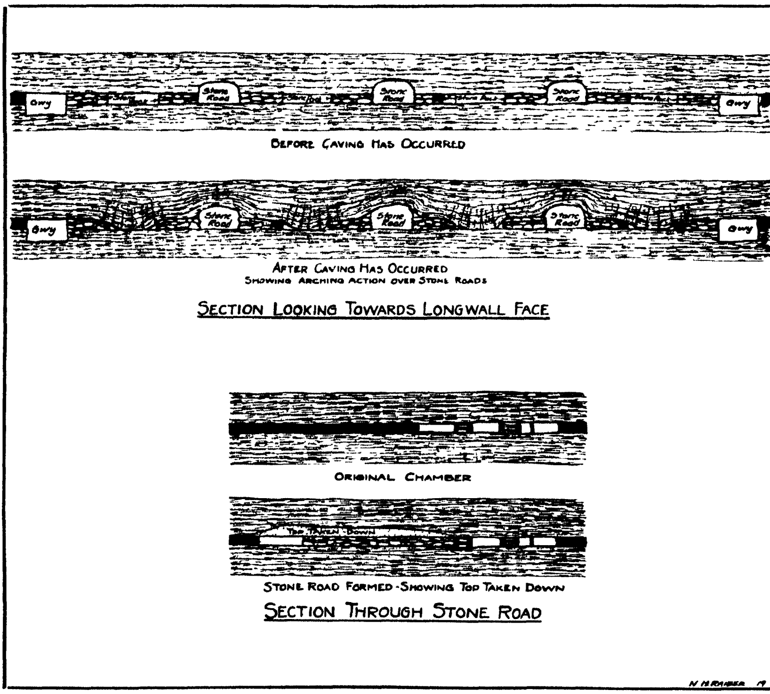

FIG. 12(*b*).—LONGWALL WITH ROCK-PACKED ROADS.

(3) It eliminates the time of placing and gathering cars in chambers. (4) It does not require as hard manual labor as hand loading does.

Pillar Robbing Operation

The scraper is used in pillar robbing in thin veins. The only difficulty, compared with hand robbing, is to keep the robbing of each pillar at an angle so that the scoop can get along the face of the pillar more readily

than if the pillar were cut off square, and also to keep the props from 6 to 7 ft. from the face of the pillar in order that the scoop may get in along the face. The scraper works under a disadvantage in this method, however, where the roof is of such a nature that it is necessary to prop closer to the face than 6 to 7 feet.

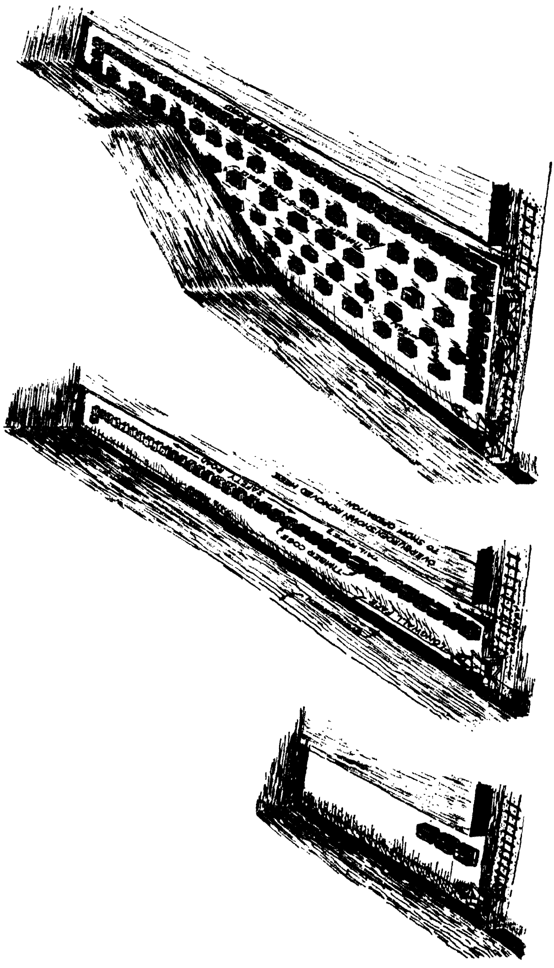

Fig 13(a).—Longwall method with timber cogging.

Longwall Work

A longwall system adapted for the anthracite region is as follows: Gangways are driven about 200 ft. (60 m.) apart and a chamber started off the lower gangway and driven up until it reaches the upper gangway.

As the chamber is being driven, a line of break props is placed and timber cogs are built in. The object of the cogs is to take the roof as it gradually settles and ease the pressure as the longwall advances. These cogs are built at intervals of 6 to 8 ft. (1.8 to 2.4 m.) in the direction the longwall face is advancing and 10 to 16 ft. in a direction parallel with the longwall face. There are two methods of supporting the roof pressure. If the

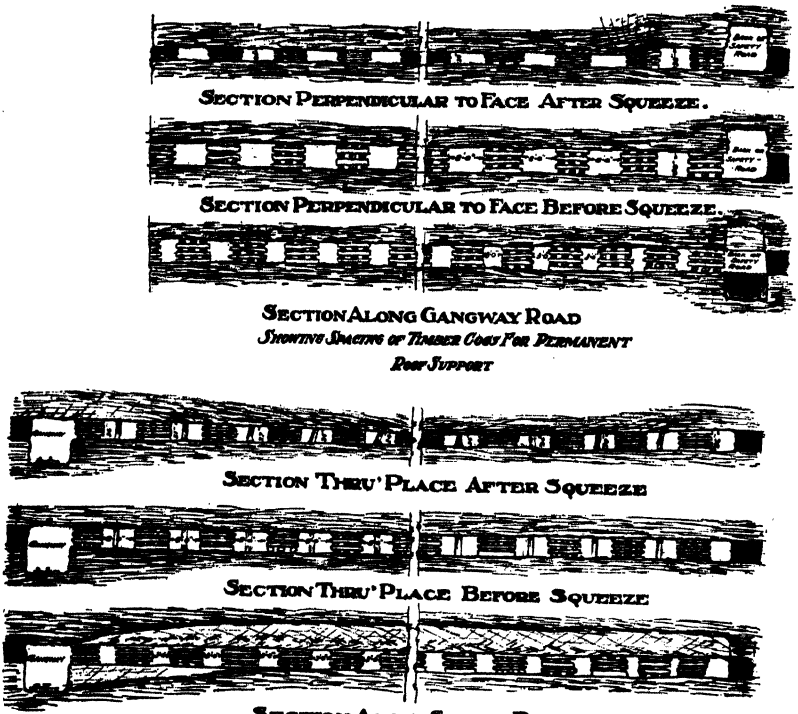

FIG. 13(*b*).—LONGWALL METHOD WITH TIMBER COGGING.

vein is less than 30 in. thick it has been found advisable to drive rock-packed roads, which also serve as safety roads, every 60 ft. and in a line parallel with the loading gangway. They are driven about 10 ft. wide and from 2 to 3 ft. of top rock is blown down. This rock is packed on both sides of the road supplying sufficient pack to a distance of 12 ft. on either side of the road. This method is shown in Fig. 12(*a*) and (*b*). Where the vein is 3 ft. or more thick, timber cogging has been used for the supports, as shown in Fig. 13(*a*) and (*b*).

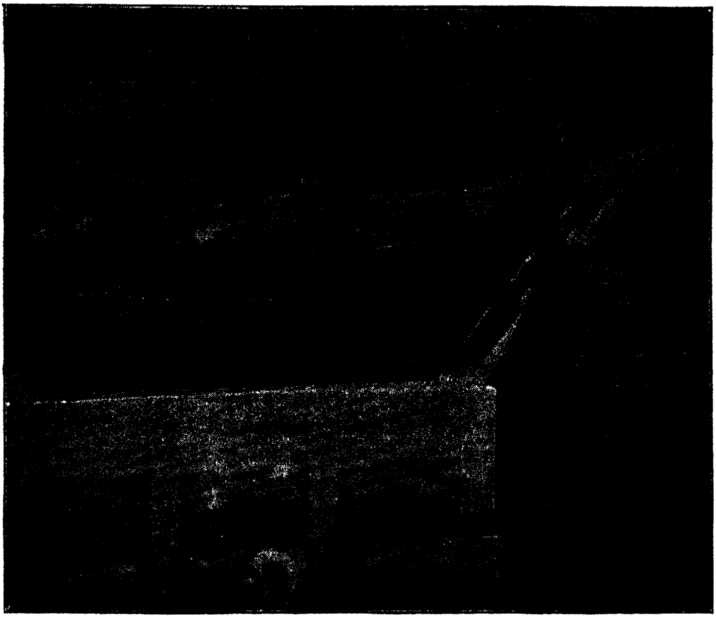

Fig. 14.—Scoop and loading apron of a longwall face. Prop at top of ladder gives a good idea of thickness of coal.

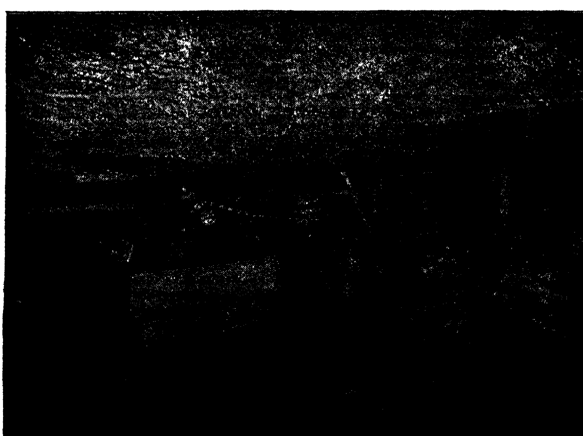

Fig. 15.—Head of conveyor line over loaded mine car.

Summary

Hand mining and loading, the car following into the chamber, is restricted by the pitch of the vein, as a car cannot be drawn up a pitch greater than 10° without unusual expense. It is also expensive to work this method in veins of less than 3½ ft. thick, because of the rock that must be removed for car clearance.

As hand drilling is slow, the output per miner is low. Transportation costs are high due to labor and supplies, cost for installation and extension of track, ties, spikes, frogs, switches, etc.; also a large transportation organization is necessary to change out cars from each chamber. A large area is required for a given output.

Hand mining and loading, using buggy, can be applied to veins thinner than 3½ ft., but not below 3 ft. and the vein must be fairly flat. As in thin veins the laborer must push the buggy while crawling on his hands and knees, it is difficult to keep men working this method in a chamber longer than 75 or 100 ft. Production per man is low due to the slow method of getting the buggy to the face and back.

Hand mining with chute loading requires a pitch of 25° on the vein. Veins as thin as 24 in. may be worked if they have the necessary pitch.

In undercutting coal and loading by hand, car following, it is necessary to drill a smaller number of holes and to use less powder to obtain a given amount of coal; it therefore, increases the output per miner. Better coal is produced. There is greater concentration of effort, which means rapid face advancement and greater output for a given area. Loaders can apply their full time toward loading coal. These advantages must overcome the labor cost of machine runner and helper, cost of undercutter, interest on the investment, and cost of repairs to the machine.

In undercutting coal and loading by scraper in room-and-pillar mining, veins as thin as 2¼ ft. (0.7 m.) may be profitably worked. The cost of taking up rock and storing it inside or outside is eliminated, also the track and track supplies in the chambers and the labor cost of installing and extending same as well as the cost of gathering cars from each chamber. A trip of empty cars can be pushed in behind the loading chute by the motor and the trip of loaded cars taken out with little delay. This method can be used to advantage in veins where coal ordinarily blows in two or three benches. But these advantages must overcome the cost of the mechanical loader and scraper equipment, interest on investment, and repairs and replacement to equipment.

Blowing coal off solid with a Jackhamer and loading by scraper can be used on pitches greater than 15°, up to chute pitch, on which an undercutter could not be operated. This method can also be used to good advantage in veins that have streaks of bone or sulfur in the bottom

6 in., which would make extremely hard cutting for the undercutter. It eliminates the machine runner and helper and the cost and upkeep of undercutter. But these advantages must overcome the following disadvantages: A greater amount of explosive is required and the miner must drill more holes for a given quantity of coal. This objection, though, is overcome to a great extent by the use of 10-ft. drill steel instead of the 6-ft. drill following the undercutter. The coal is blown into smaller pieces than undercut coal, and hence, greater difficulty is experienced in building topping on the cars.

The longwall method using undercutter and loading with scraper allows the undercutter to make a straight cut of about 200 ft. in 8 hr. without withdrawing the cutter arm. A greater tonnage may, therefore, be undercut in a shift than if the machine is moved from one chamber to another. The longwall machine can cut both ways, thus eliminating dead traveling. As this coal is easily loaded out by the scraper with very little changing of pulleys and with no handling of the rope, the output per mechanical loader is about twice that of a chamber operation. The output per machine runner, miner, and scraper-man is greatly increased and the maintenance charges on mechanical equipment are reduced because of the increased tonnage per machine.

As all the coal is taken out at once it is unnecessary to go back to rob pillars. In thin-vein mining, robbing pillars with the mechanical loader is a more costly operation than first mining. Veins as thin as 24 in. (60 cm.) may be worked profitably and it is possible to work veins as low as 20 in. with the mechanical equipment now in use.

Longwall advancing yields an immediate return on the investment.

In longwall retreating, transportation roads are maintained in solid coal instead of being in disturbed area. As the face retreats, the roads behind can be forgotten and the track removed. The expense of keeping roads open is eliminated. This method is safer for men in case of a general squeeze. It is suitable to weak roofs but involves an outlay of capital in development work.

The most important factor in the efficient and continuous operation of any mechanical contrivance used for mining or loading coal is a good and sufficient supply of cars to take away the coal as quickly as it can be put into them. To maintain a maximum output with the scraper, the work and the working force must be so arranged that the scraper will be continuously traveling up and down the chamber. As a car must be under the loading platform at all times, the empty cars should be taken through the airway, or some other road, and brought back in the rear of the loading platform, then run past the platform as they are required for loading. A slight grade in favor of the loaded cars is an advantage.

Statistics

	Undercutting and Hand Loading		Hand Mining and Loading	
Thickness of vein, inches	60	61	60	61
Thickness of coal, inches	40	50	40	50
Men employed	28	13	14	61
Undercutters in use	2	1		
Tons produced per day	138	62	43	186
Tons per man per day	5.0	4.8	3.1	3.0
Tons per miner per day	35.0	31.0	4.8	6.2
Tons per undercutter per day	69	62		
Operating cost per ton	$ 1.37	$ 1.30	$1.87	$1.64
Maintenance cost per ton	0.16	0.18		
Depreciation of equipment per ton	0.03	0.04		
Total cost per ton	1.56	1.52	1.87	1.64

	Chamber Undercutting and Scraper Loading			Hand Mining and Loading		
Thickness of vein, inches	33	30	49	33	32	42
Thickness of coal, inches	26	30	42	27	32	42
Men employed	18	18	9	22	12	18
Undercutters in use	2	2	1			
Scraper loaders in use	2	2	1			
Tons produced per day	48	67	35	59	32	50
Tons per man per day	2.7	3.7	4.0	2.7	2.7	2.8
Tons per miner per day	10.3	16.0	14.5	4.2	5.3	4.1
Tons per undercutter per day	24	33	35			
Tons per scraper loader per day	24	33	35			
Operating cost per ton	$ 2.30	$ 1.92	$ 1.77	$2.70	$2.50	$1.82
Maintenance cost per ton	0.20	0.17	0.20			
Depreciation of equipment per day	0.10	0.12	0.11			
Total cost per ton	2.60	2.21	2.08	2.70	2.50	1.82

	Jackhamer Mining and Scraper Loading				Hand Mining and Loading		
Thickness of vein, inches	29	33	47	42	32	47	42
Thickness of coal, inches	29	33	36	42	32	36	42
Men employed	8	7	18	11	9	51	18
Jackhamers in use	1	1	3	2			
Scraper loaders in use	1	1	3	2			
Tons produced per day	28	26	86	85	32	132	50
Tons per man per day	3.5	3.7	4.8	7.7	3.5	2.6	2.8
Tons per miner per day	14.7	13.4	14.3	21.8	5.3	4.3	4.1
Tons per Jackhamer per day	28	26	29	42			
Tons per scraper loader per day	28	26	29	42			
Operating cost per ton	$ 2.33	$ 1.95	$ 1.48	$ 1.28	$2.50	$1.96	$1.82
Maintenance cost per ton	0.06	0.10	0.12	0.05			
Depreciation of equipment per ton	0.06	0.06	0.05	0.04			
Total cost per ton	2.45	2.11	1.65	1.37	2.50	1.96	1.82

LONGWALL MINING WITH UNDERCUTTER AND SCRAPER

	Using Rock Pack		Using Timber Cogs		Hand Mining and Loading		
Thickness of vein, inches	24	29	36	48	27	36	48
Thickness of coal, inches	24	22	36	48	27	36	48
Men employed	10	15	20	8	2	20	16
Undercutters in use	1	3	2	1			
Scraper loaders in use	1	3	2	1			
Tons produced per day	34	83	136	64	2	48	52
Tons per man per day	3.4	5.5	6.8	8.0	1.0	2.4	3.2
Tons per miner per day	11.2	16.7	21.2	32.0	2.0	4.4	5.1
Tons per undercutter per day	34	28	68	64			
Tons per scraper loader per day	34	28	68	64			
Operating cost per ton	$3.00	$2.50	$1.75	$1.44	$3.90	$2.15	$1.72
Maintenance cost per ton	0.17	0.25	0.15	0.06			
Depreciation of equipment per ton	0.17	0.15	0.05	0.05			
Total cost per ton	3.34	2.90	1.95	1.55	3.90	2.15	1.72

SIZE OF CUTTINGS MADE BY MINING MACHINE

	Per Cent.
Stove	6.0
Chestnut	18.0
Pea	28.0
No. 1 Buckwheat	21.0
No. 2 and No. 3 Buckwheat	19.0
Smaller	8.0
	100.0

From a study of the foregoing statistics the following conclusion may be drawn:

1. *Value of Mechanical Mining in Thin Veins.*—In nearly every case, an economy results from the introduction of mechanical mining. Veins down to 24 in. thick may be worked economically by means of mechanical devices; whereas hand mining, with the possible exception of a few cases, cannot be economically carried on in coal as thin as this. This results in a considerable increase of workable coal areas in the northern anthracite fields, and tends to prolong the life of the industry and the conservation of natural resources.

2. *Increased Production per Miner.*—The production per certified miner from a mechanical-mining operation is from two to six times as much as from the hand-mining system; therefore per given production, fewer miners are required. This point has particular value because the miner is a skilled or high-priced worker; also, the Pennsylvania state mining laws require that a miner must have at least two years' experience as a mine laborer before qualifying as a certified miner, which has a tendency to restrict the possible number of miners in the region. In addition, the work of the miner is less laborious when mechanical devices are used.

3. *Greater Output from Thin Veins.*—For a given area or a given amount of development, a far greater production per day is obtained from mechanical mining than from hand mining, due to the concentration of effort on a small area. This also facilitates the transportation and the incidental mine work. A steadier production is obtained from mechanical-mining units, as the coal is loaded out with few interruptions, while in hand mining a certain number of days in every period are employed in blowing and loading out or gobbing rock, which means the non-production of coal. By means of cross-over connections between gangway and airway, a number of mechanical mining units may be worked in the same section of the mine without interfering with one another, thus considerably increasing production.

4. *Greater Output per Employee.*—The output per employee for mechanical-mining operations is about 50 per cent. greater in chamber work than hand chamber methods, and on longwall work as high as 300 per cent. increase has been obtained. This, in itself, is a very important point in the favor of mechanical methods.

5. *Greater Prepared and Total Yields.*—Tests show that coal undercut by machine gives about 10 per cent. more prepared yield than hand mined coal. Jackhamer-mined coal blown off the solid does not give as good a prepared yield as hand-mined. As the speed of production is greater in the Jackhamer method, the whole face is drilled at one time and fired instead of drilling only a few holes at a time, as in hand work. This does not permit the greatest care in the placing of holes and providing free breaking faces, as in hand mining.

6. *Percentage of Recovery.*—By mechanical-mining methods, where chamber-and-pillar work is done, the same amount of recovery can be made as made in hand mining; in the longwall system of mechanical mining, 100 per cent. recovery is being made.

In conclusion, it may be said that the advantages of mechanical mining, as applied to thin veins, are much greater in the case of thick veins. Generally speaking, mechanical mining has only been introduced in thin-vein work, or in thick-vein work where the physical conditions were abnormal and where it was impossible to secure men to work by the hand method. For this reason, there has been little opposition to the mechanical-mining method from the labor unions. However, when the question of introducing mechanical mining into the thick seams is to be considered, as with the introduction of any machinery to replace hand work, means of controlling any possible opposition on the part of the workman must be taken into consideration. Furthermore, it is readily seen that the mechanical methods of mining will produce the same amount of tonnage that hand methods of mining will produce with a reduction of about 33 per cent. in producers, and a like reduction in company forces due to the concentration of transportation, working faces,

etc. If a virgin area of coal is to be worked, due consideration should be given to the introduction of mechanical methods for the entire mining of this coal, which undoubtedly under normal working conditions will result in quicker extraction, increased output per employee, and quicker returns on investment.

I wish to thank Mr. A. J. Wiegand for the assistance he rendered in preparing this paper.

DISCUSSION

Cadwallader Evans, Jr.,* Scranton, Pa.—Would it be possible, when this paper is published in the Transactions, to include some information about the rates of pay that governed when these figures were made up? The costs are given in dollars and cents per ton, but it would be much more illuminating if we had the basis on which the men were paid at that time.

Herbert D. Kynor.—These figures are based on 1921 wage rates.

Sidney J. Jennings,* New York, N. Y.—In the maximum year for the mechanical mining of anthracite, the total output is given as 2,000,000 tons; what proportion does that 2,000,000 tons bear to the total amount produced by the companies that use coal-mining machines?

Herbert D. Kynor.—In the case of the Hudson Coal Co., in that year about 5 per cent. of the production, which was over 7,000,000 tons, was produced by machines. I do not know the proportion in the case of the other companies because I do not know their total tonnage figures.

W. J. Richards,† Pottsville, Pa.—The Jackhamer was included among the mechanical instruments; the 2,000,000 tons does not cover the tonnage produced by the aid of the Jackhamer?

Herbert D. Kynor.—It covered the tonnage where the coal was mechanically loaded.

Geo. S. Rice,‡ Washington, D. C.—Why is there an apparent falling off in the production of mechanical mining and loading, where perhaps we might expect a continued gain?

Herbert D. Kynor.—The falling off was due to the labor troubles of the Pennsylvania Coal Co. last year; a four-months strike reduced its tonnage materially.

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* Consulting Engineer.

† President, Philadelphia & Reading Coal and Iron Co.

‡ Chief Mining Engineer, U. S. Bureau of Mines.

Skip Hoisting for Coal Mines

By Andrews Allen, C. E., and John A. Garcia, E. M., Chicago, Ill.

(New York Meeting, February, 1921)

The large increase in the wages of mine workers makes it imperative that all factors tending to limit production per miner be eliminated, if possible. The trolley and storage-battery locomotive, mining machine, hoisting engine, mining system, etc., have increased the possible output per man. But this output is still limited by the capacity of the hoisting engine and cable and by the size of the mine car.

The superiority of skip hoisting in metal mining is shown by its almost universal adoption. By varying the size of skip and the rope speed, any desired tonnage can be secured and, since frequently only one kind of material is handled and breakage is unimportant, the loading of the skips can be easily and cheaply effected from bins into which the cars are dumped. The cars may then be designed to fit the conditions in the mine instead of being a compromise between hoisting and mining conditions, very likely suiting neither.

In coal mines, the usual practice has been to hoist the car to the surface, either on platform or self-dumping cages. Of late years the number of skip-hoisting plants in coal mines has been rapidly increasing, but there is more or less inertia to overcome in establishing so radical a change in practice; also the earlier skip operations were not uniformly successful. As a result, it became evident that metal-mining practice would require radical modification before skip hoisting could be successfully used in coal mining.

Objections to Skip Hoisting

Breakage

Except where coal is used for coking, or for other purposes where breakage is unimportant, breakage of the coal is vitally important all the way from the face to the railroad car. The average selling price is seriously reduced when the percentage of screenings is increased. Following metal-mining practice exactly, the coal would be dumped into a deep pocket, cut off into a skip load by a measuring hopper or other means, dumped into a deep, narrow skip, and dropped into a bin at the top, all of which would contribute to excessive breakage.

It is difficult to make a direct comparison between the breakage of coal in mines operated with self-dumping cages and with skips. The amount of fines is influenced by other considerations than handling; for instance, the character of the coal and the percentage of narrow work that is being done at the time the test is made.

The complete record of the percentages of two Illinois mines that were started about the same time and are producing about the same tonnage is available. In one, the Kathleen mine of the Union Colliery Co., the coal is hoisted in 10-ton skips; in the other, the No. 2 mine of the J. K. Dering Coal Co., at Eldorado, the coal is being hoisted in 3½-ton cars. The record for 1920 up to Oct. 1 for these two mines is as follows:

J. K. Dering Coal Co., Cage-hoisting Mine

	Per Cent.
6 in. lump	9.3
6 × 3 in. egg	20.4
1¼ × 3 in. nut	21.6
1¼ in. screenings	48.7
Total	100 0

When the 2 in. (50.8 mm.) screenings are considered, the proper proportion of nut added to the 1¼ in. (31.75 mm.) screenings gives 54 per cent for the same period.

Kathleen Mine, using 10-ton Skips

Size	October, Per Cent.	Jan. 1. to Oct. 31, Per Cent.
2 in. lump	2.4	10.4
3 in. lump	0.1	
6 in. lump	12.7	10.5
3 × 6 in. egg	16 2	14.4
2 × 3 in. nut	18.0	13.9
2-in. screenings	50.6	50.8
Total	100 0	100.0

It will be noted that the percentage of the 2-in. (5 cm.) screenings is less in the case of the skip-hoisting mine than in the cage operation, which bears out the theory that handling coal in large masses in skip hoisting does not cause any more breakage than does the handling of coal on self-dumping cages.

Bottom Installation for Dumping Cars and Loading Skips

The cost of such an installation is greater than an ordinary coal-mine bottom, where cars are run directly on to a platform or self-dumping

cage. However, many important savings may be effected when the entire cost of the plant is considered, and the expenditure may be justified by the greatly increased hoisting tonnage and the decreased cost of labor.

Dust

As coal dust is inflammable, the dumping of coal at the bottom of a downcast hoisting shaft might be considered a source of danger. This objection, in the writers' opinion, has been greatly overestimated. From a close observation, they think that only an extremely small amount of dust so produced finds its way into the entries. There are effective means of taking care of dust where it may be a serious matter. Where the hoisting shaft is upcast, there is no trouble from this source.

Difficulty of Inspection and Docking

This, in the opinion of the writers, is one of the most serious objections to skip hoisting, yet it is easily overcome. In a large mine it is impossible to examine and dock each car separately. The effort of mine management should be directed to securing clean coal at the face, and the only practicable method of checking the loaders is to employ a "spotting system," by means of which a certain number of cars, taken at random or from sections of the mine where careless loading is suspected, can be taken out, carefully and thoroughly inspected, and the docks identified and charged against the guilty loader.

Where the coal is dumped indiscriminately into a deep pocket, no docking or spotting system is possible, except through the use of an auxiliary shaft; but where a skip installation is properly designed, it is just as possible to dock with a skip as with a cage, and without any greater inconvenience or sacrifice of output.

Handling of Rock

Many coal mines have to hoist considerable quantities of rock; where cages are employed, the rock can be taken as it comes in the trip and handled at the top by means of a fly gate or some similar method. Where skips are loaded from a pocket, rock can be handled only at an auxiliary shaft, or in a second shift at the main shaft. Both methods require cutting the rock cars out of the trips, which is easily possible where care is taken in assembling the cars at the partings.

It is possible in a skip mine to provide an excessive hoisting capacity, which will take care of the rock as well as the coal; where the deep pocket is not used, the design can be easily arranged so that a moderate quantity

of rock can be handled at the main hoisting shaft without the slightest difficulty.

Handling of Men and Materials

A skip will not handle men or material unless equipped with an auxiliary deck, in which case it becomes cumbersome and inconvenient. Even this can be done successfully, but the interruptions of hoisting, due to handling men and material during the day shift, are serious in any large mine; practically all modern, large-tonnage mines have found it necessary to put an auxiliary hoisting plant in a separate compartment, usually in connection with the air shaft. The use of such an auxiliary hoist answers this objection completely.

Advantages of Skip Hoisting

1. A hoisting capacity, capable of taking care of all the coal and rock that can be mined.

2. A smaller shaft.

3. A large ratio of lading to gross weight. For example, for an 8-ton skip and a 4-ton self-dumping cage this ratio is:

	Skip Installation	Cage Installation
Weight of cage or skip	13,000	14,000
Weight of car		4,000
Weight of coal	16,000	8,000
Total	29,000	26,000

In the case of the cage, the coal constitutes about 30 per cent. of the total weight hoisted, while in the case of the skip, the coal is 55 per cent.

4. A low rope speed, resulting in lower power consumption, and a less expensive hoisting plant, greater regularity and safety in hoisting, including freedom from the usual hoisting casualties, which are the result of the speed and violence of cage operation.

5. Entire absence of hoists that "miss the hook," or cars that go into the basket in self-dumping operations, which result in wrecks and delays.

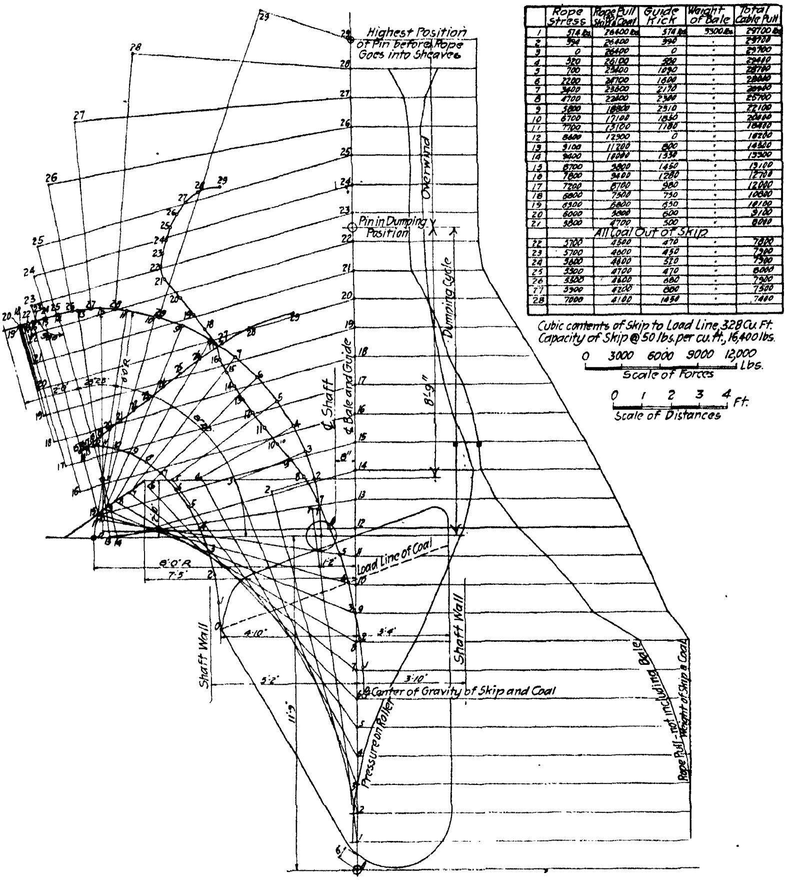
6. Requires not more than one-half the men ordinarily employed for a smaller self-dumping cage.

7. Use of solid-end cars instead of cars with lifting or hinge-end gates. This reduces the first cost of the car, cuts the maintenance cost in half, and prevents dust and spillage of coal on the roadway.

THE SKIP

Bottom-dumping Type

This type of skip is rectangular, of uniform cross-section, and of any required depth. The bottom slants at a steep angle to the gate in one side, which is operated by engagements in the tipple and held, when



	Rope Stress	Rope Pull Skip & Coal	Guide Kick	Weight of Bale	Total Cable Pull
1	574 lbs.	26400 lbs.	574 lbs.	5500 lbs.	29700 lbs.
2	594	26400	390	"	29700
3	0	26400	0	"	29700
4	520	26100	580	"	29400
5	700	25400	1090	"	28700
6	2200	24700	1600	"	28000
7	3400	23600	2170	"	26900
8	4700	22400	2300	"	25700
9	5800	18800	2310	"	22100
10	6700	17100	1850	"	20400
11	7700	15100	1180	"	18400
12	8600	12900	0	"	16200
13	9100	11700	800	"	14500
14	9400	10000	1330	"	13300
15	8700	9800	1450	"	13100
16	7800	9400	1280	"	12700
17	7200	8700	980	"	12000
18	6800	7500	730	"	10800
19	6300	6800	650	"	10100
20	6000	5800	600	"	9100
21	5800	4700	500	"	8000
	All Coal Out of Skip				
22	5700	4500	470	"	7800
23	5700	4600	450	"	7200
24	5600	4600	510	"	7500
25	5500	4700	470	"	8000
26	5500	4600	680	"	7900
27	5300	4200	800	"	7500
28	7000	4100	1450	"	7400

FIG. 1.—A. & G. SKIP DIAGRAM.

running in the shaft, by means of a latch or toggle levers. The best example of this type is known as the Lepley skip. The gate is held in place by toggle levers and, when opened, by engagement with the tipple, acts as a chute extending over the dump plate.

In the case of an 8-ton skip, the discharge lip must be hoisted about 2

ft. 6 in. (76.2 cm.) above the point at which discharge of coal is commenced before the final position of the skip is reached. At this point the skip may stand stationary while the coal flows out, which should not take more than 4 or 5 sec., during which time the other skip is being filled.

The objections to a skip of this character are: (1) The mechanism must be attached to the skip; this increases the weight and cost and puts the working parts in a position where they are easily damaged but are difficult to repair or lubricate. (2) The hinged gate is a weak point. The coal dumps directly upon it and, like the operating mechanism, it is subject to serious wear that, in time, will cause leakage. (3) The form of the skip makes it necessary to dump the coal directly into a V-shaped bottom, which produces more breakage than any other form.

The advantages are: (1) A slightly shorter hoist during the dumping cycle. In the Lepley 8-ton skip, the loading lip will be about 14 ft. 6 in. (4.42 m.) above the dump chute when the final dumping position is reached, while with the overturning skip of same capacity, this distance is about 4 ft. 6 in. (1.37 m.) greater. (2) Greater capacity in a given size of shaft, compared with the round-bottom type, as the skip may be made as deep as desired. (3) Adaption to installation of a platform for men or materials. (4) A larger dead weight available for counterweighting the loaded skip when it is lifted off the bottom. The entire weight, 17,000 lb. (7.7 T.) is available for counterweighting, while a round-bottom overturning skip weighing 13,000 lb. (5.9 T.) will lose about 5000 lb. (2.26 T.) leaving only 8000 lb. (3.63 T.) available for counterweighting the loaded skip.

Overturning Skip

The type of overturning skip that has been successfully adapted to coal mining is much like an old-fashioned coal scuttle in shape, with a rounded bottom and a discharge side about 30° off the vertical. This form saves breakage because the coal never hits a square blow on the steel side or bottom; as the skip is not excessively deep, the slanting discharge side makes it possible to discharge the coal completely with a dumping angle of 105° in place of the 135° required by a rectangular skip.

The cycle of operation is shown in Fig. 1. It will be noticed that the discharge of coal commences at point 8 and is finished at point 21, covering a vertical hoisting height of 13 ft. (3.96 m.) commencing well below the dumping point and finishing at a point where the loading lip of the skip, if hoisted vertically, would be 19 ft. above the dumping point. During this hoisting distance, the coal is discharged uniformly, and sustains little drop when flowing on to the chute.

The skip shown is loaded from the side opposite its discharge; in this way the greatest capacity is obtained. The depth of skip from loading

lip to bottom is fixed by the width of the shaft and the angle of the discharge side. The coal naturally piles up toward the discharge point on a line approximately as shown on the diagram.

The skip is held vertical in the shaft by gravity, being pivoted slightly in front of the center of gravity; also by a spring latch on each side which is opened by an engagement on the tipple and automatically snaps into position when the skip descends.

	Self Dumping Cage	Overturning Skip	Bottom Dumping Skip
Depth of shaft	400'	400'	400'
		Pit 24'	Pit 24
	Tipple 60'	Tipple 46'	Tipple 46'
Total hoist	Dumping 6' = 466'	Dumping 19' = 489'	Dumping 15' = 485'
Capacity	800 ton per hr	800 ton per hr	800 ton per hr
Hoisting cycle: Loading or caging	4 sec	8 sec	8 sec
Accelerating	6 sec	8 sec	8 sec
Running	4 sec	15 sec	15 sec
Retarding	4 sec 18 sec	5 sec 36 sec	5 sec 36 sec
Weight of cage or skip	14000 #	13000 #	17000 #
Car	4000 #	—	—
Coal	8000 # - 26000 #	16000 # - 29000 #	16000 # - 33000 #
Size of hoisting rope	1½"	1⅝"	1⅞"
Hoist during dump	6-0"	13-0"	2-6"
Net wt available for ctwt	12000 #	8400 #	17000 #
Average rope speed	2000' per minute	1030' per minute	1040 per minute
Max rope speed	3610' " "	1480' "	1475 " "
Size of cylindro conical drum	7' to 11' diam	8' to 11' diam	8' to 11' diam
	Direct connected	Geared	Geared
" D.C. hoist motor (sq root mean square)	2000 HP	850 HP	843 HP
Size of D.C. generator on Illgner set	1400 KW	600 KW	600 KW
" " A.C. motor on Illgner set	800 HP	600 HP	600 HP
A.C. input per hour	656 KWH	525 KWH	516 KWH
" " per ton of coal	0.82 KWH	0.657 KWH	0.650 KWH
Hoist overall efficiency based on 466' hoist	43.0%	53.8%	54.3%
Approx cost F.O.B. factory (1920 basis) Drum	$25000.00	$30000.00	$30000.00
Electrical equip't	$90000.00 $115000.00	$48000.00 $78000.00	$48000.00 $78000.00

FIG. 2.—COMPARATIVE DATA FOR CAGE AND SKIP HOISTS.

The dumping is effected by two well lubricated wheels of comparatively large diameter at the top of the skip engaging long angle-iron circles carefully formed to the exact dumping curve. The skip is held rigidly between the bales by tapered plates, which bring it into a true vertical position as the skip returns into the shaft.

It is possible to omit the latches and to place a second set of guides in the shaft, so that the dumping wheels will come in contact with them if the skip should become unbalanced. Both of these types are in successful operation.

The objections to this skip, compared with the bottom-dumping type, are: (1) A slight sacrifice of capacity, meaning a slightly larger shaft for the same capacity. (2) A slightly greater hoist to accommodate the dumping cycle at the top, about 4 ft. 6 in. (1.37 m.), for an 8-ton skip. (3) Less load on the hoisting rope available for counterweighting the loaded skip when the hoisting cycle has been completed.

The advantages of this type are: (1) Simplicity and ruggedness; (2) lightness in proportion to the load; (3) less breakage; (4) an easy and

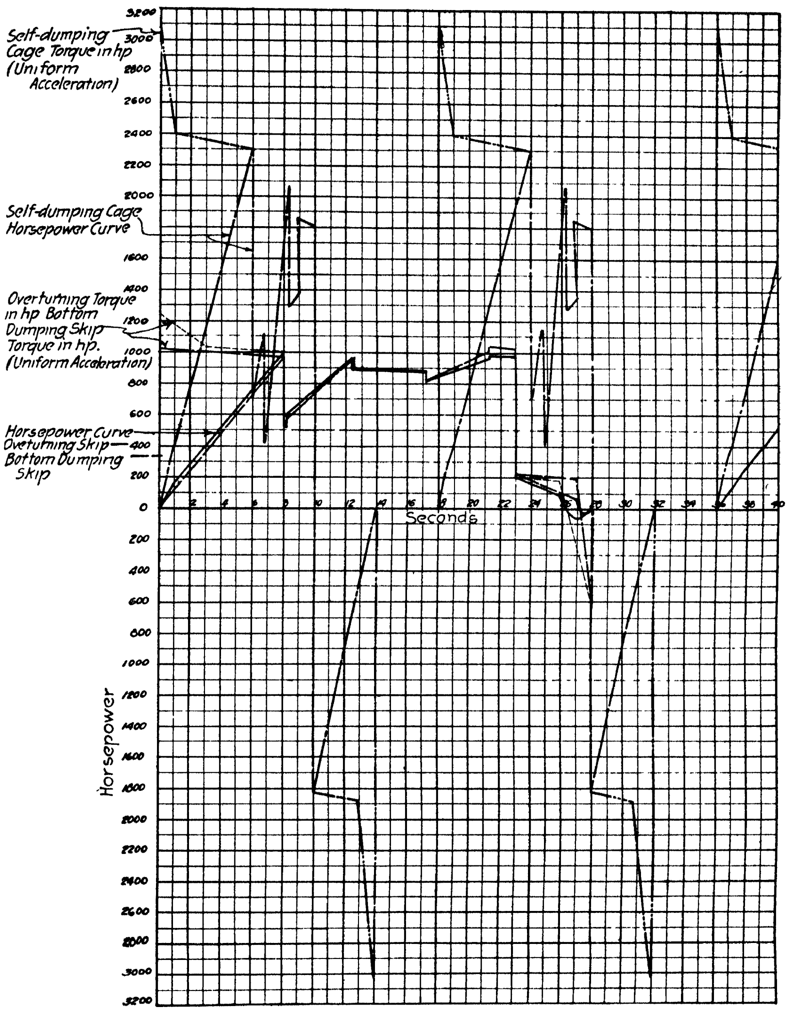


FIG. 3.—POWER CONSUMPTION CURVES.

uniform discharge without the possibility of large pieces of the bottom-lodging in the gate and with no more drop than in the case of the bottom-dumping skip.

Hoisting Capacity

A skip can be made to handle almost any desired capacity; those now in use, with which the writers are familiar, handle from 5 to 13 tons maximum for balanced hoisting. A 50-ton skip is being designed for counterweighted operation. A skip should be designed for some multiple of the mine-car capacity; then the cars can be dumped directly into it, eliminat-

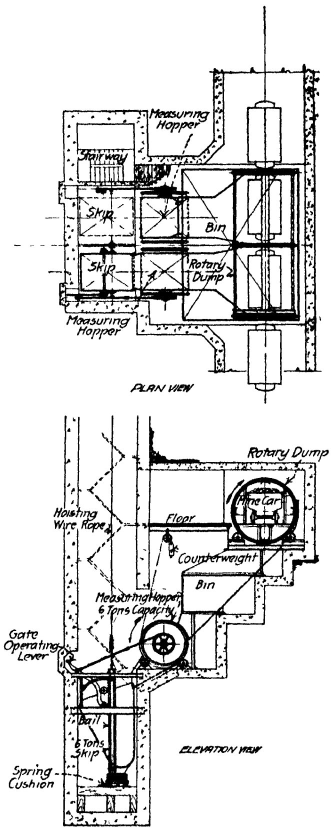


Fig. 4.—Pocket and measuring hopper diagram.

ing the deep pocket and measuring hopper. The time required to load one of these skips will average from 5 to 8 sec. measured from the time the skip lands on the timbers to the time it is belled away. With balanced hoisting, an average load of 9 tons, a total hoist of 400 ft. (121.9 m.), and an average rope speed of 1000 ft. (304.8 m.) per min., it is possible to obtain a capacity slightly over 1000 tons per hour.

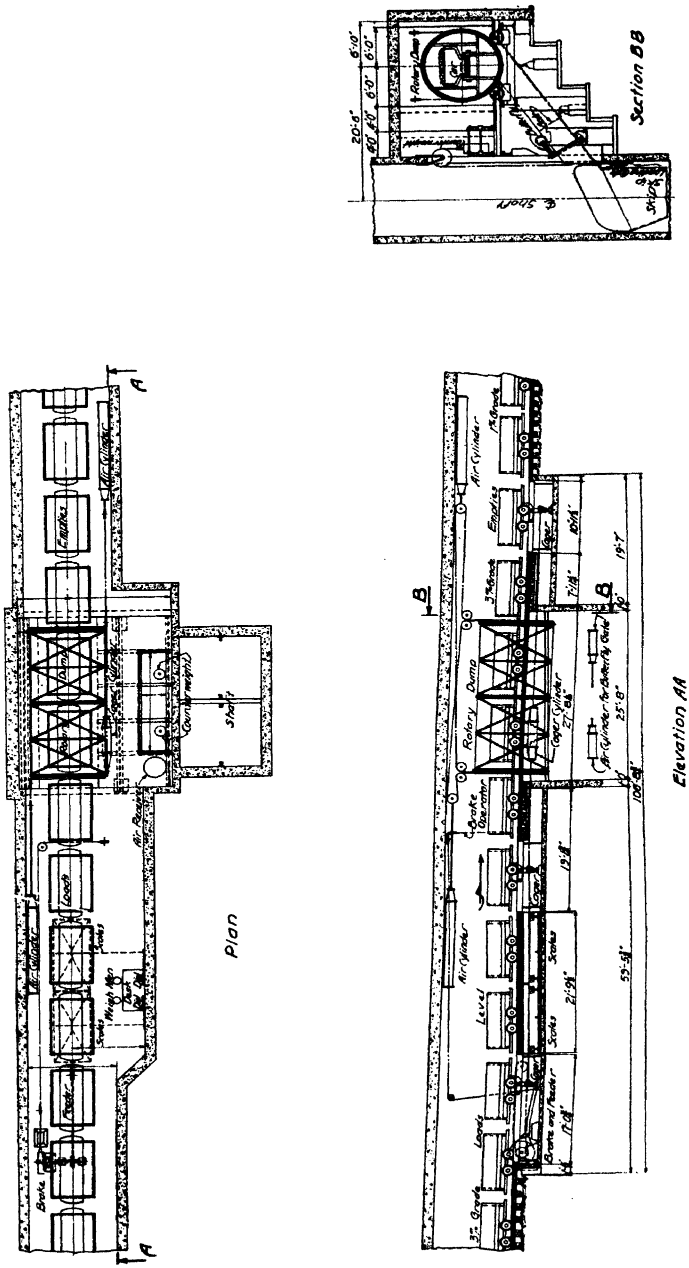


FIG. 5.—KATHLEEN MINE LOADING STATION.

Power Consumption

Figs. 2 and 3 give a comparison between the power consumption, motor sizes, and cost of hoist for three assumed installations designed for the same capacity. The first is a self-dumping cage installation; the second, an installation using overturning skips; and the third, an installation of the Lepley type. The comparisons are made on the basis of a

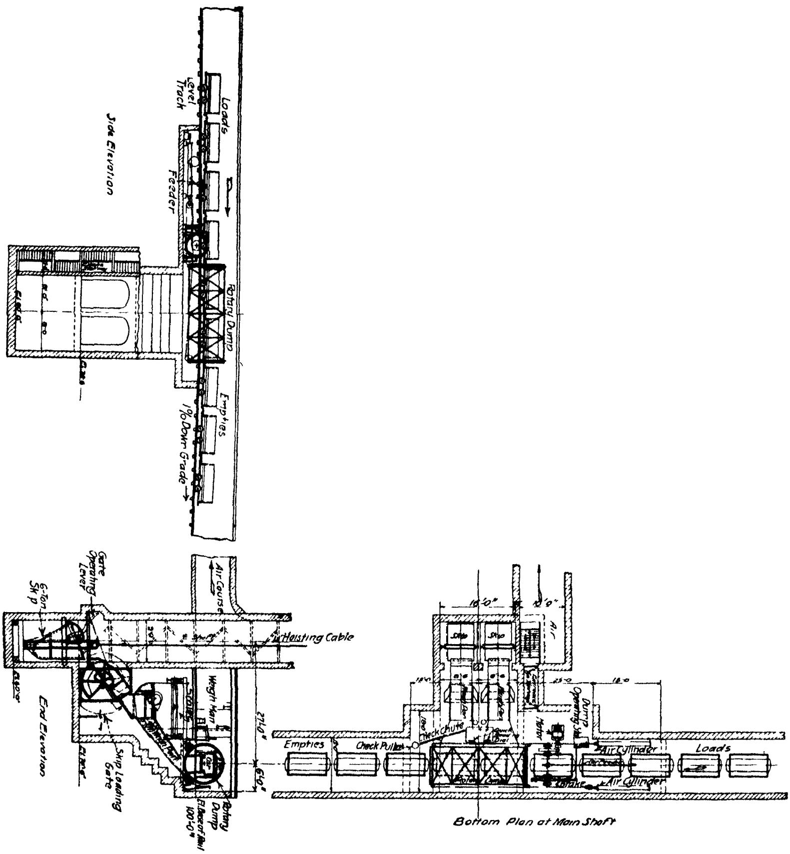


Fig. 6.—Rosita, Mexico, mine loading station.

400-ft. shaft and a hoisting capacity of 800 tons per hour. This, of course, is somewhat high for cage hoisting, although it is within the limits of actual practice. If the comparison had been made on the basis of 1000 tons per hour or for a deeper shaft, the skip installation would show only a moderate increase in power consumption, while the cage installa-

tion would increase at such a rapid ratio as to soon exceed the practicable limits of construction and operation.

The comparisons could not be made for three existing mines of these types, as it would be impossible to get exactly parallel comparisons. The figures used, however, are based on actual results and are worked out exactly on the same basis.

Further, the comparison is made on the basis of direct-current hoisting with an Illgner flywheel set, so proportioned as to take the peak load off the line, thus giving a uniform power consumption for the cycle assumed. If the peak load is not too great for the transmission line, an alternating-current hoist may be used, which will show a large economy in first cost. It will be noted that the peak of the cage-hoisting cycle is about 3100 hp. (2311.7 kw.), while the peak of the skip-hoisting cycle is a little over 1000 hp. (745.7 kw.) for the bottom-dumping skip and a little over 1200 hp. for the overturning skip, the motor sizes being approximately 2000 hp. in the case of the cage and 850 hp. in both skip installations. Neither of these, under ordinary circumstances, would permit the use of an alternating-current hoist, but an alternating-current hoist can be used to a much larger capacity with the skip than with the cage.

Where steam hoists are used, there is no such difference in favor of the skip, as the size of the hoisting engine is determined by the total load rather than by the acceleration peak. The steam consumption will, however, follow approximately the power curves of the diagram.

The diagrams and tables show that either skip installation has a power economy of nearly 25 per cent., compared with the cage; also there is a difference in cost of approximately $37,000 in the examples cited.

Bottom Loading Station

Where no attention is paid to breakage and where inspection of coal and handling of rock are provided for elsewhere, there are some advantages in the bottom layout in which the cars are dumped directly into a deep pocket, which can be used to store a large amount of coal. From the pocket the coal flows into an automatic measuring hopper, and then to the skip by means of a gate operated by the skip itself. This arrangement is shown in Fig. 4.

This method has two advantages: the skips can be loaded to full capacity irrespective of the size and loading of the cars, and the dumping operations can be carried on without reference to the hoisting cycle, thus reducing the number of cars used in the mine, but a few more pit cars will provide greater storage at less cost. In an outfit of this kind, it may be possible to secure the desired dumping capacity with a single-car rotary dump, instead of the two-car rotary dump, which is necessary where the dumping is regulated by the skip, though at the cost of breakage, and inability to dock and to handle rock.

The proper arrangement for a coal mine should involve dumping the coal as directly as possible from the cars into the skip, which must then be some multiple of the car capacity. Where cars are of moderate size and the shaft is not too deep, it is usually convenient and economical to use a skip that will hold two carloads of coal. With a two-car rotary dump, two arrangements can be employed. In one, the contents of both cars are thrown to each skip alternately; in the other, each car

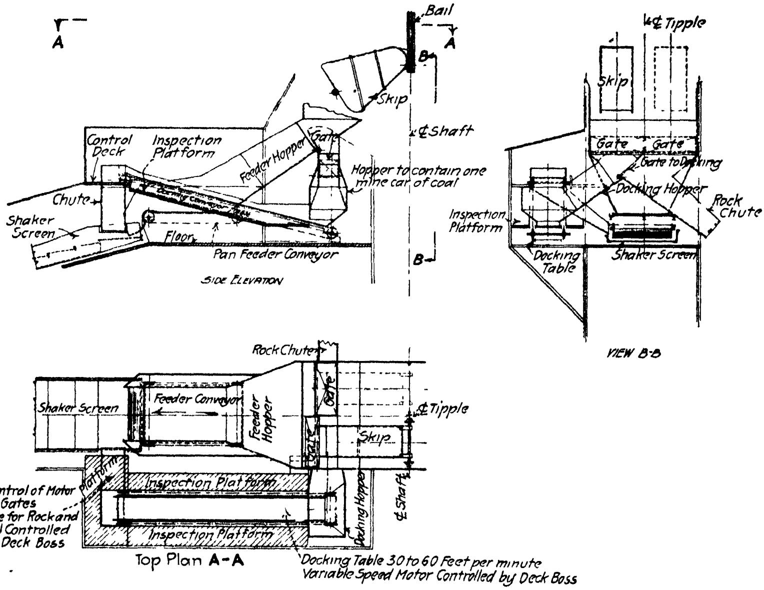


FIG. 7.—DOCKING TABLE ON TOP.

dumps into a separate weigh pan or chute, the second dump going on top of the first, thus providing two carloads of coal for each skip load when operations are started with a single dump in one skip. The first arrangement is shown in Fig. 5 and second in Fig. 6. This arrangement eliminates the deep pocket, reduces breakage, provides facilities for docking and for handling rock, and seems to meet the conditions and requirements for coal mining.

THE CAR DUMPER

For dumping the cars, four arrangements are used: the cross-over dump, the kick-back dump, the bottom-dumping car, and the rotary dump. The first two types require cars with end gates, which are of course objectionable. These dumps are inexpensive, require no power, and are easily installed. The cross-over dump is rapid. None of these dumps, however, can operate in multiple or handle coupled trips.

The kick-back dump can be easily adapted to skip hoisting, one of these dumps being located in front of each shaft compartment. In this way skip hoisting may be installed in an existing mine without any radical reconstruction of the bottom.

It should be noted that the mine bottom for skip hoisting runs at right angles to the mine bottom designed for cages. There seems to be no other convenient or economical way of adapting an existing mine bottom to skip hoisting without driving new entries.

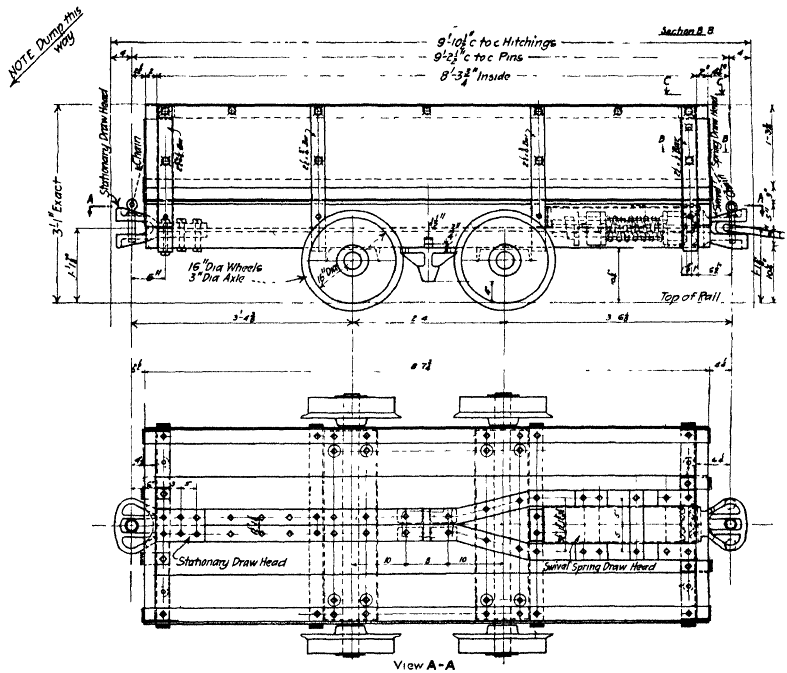


FIG. 8.—TYPE OF MINE CAR FOR SKIP MINE.

A thoroughly satisfactory bottom-dumping car has not been designed, although many types are in operation. All require mechanism on the car itself; the cost of which would be great if not prohibitive. In addition, bottom-dumping cars lack solidity and strength. They sometimes dump in the mine, causing wrecks; they spill more coal dust than cars with end gates; they are expensive to maintain, and frequently do not clean themselves readily, large lumps having a tendency to stick in the cars unless an extra wide track gage is used. If a satisfactory and economical bottom-dumping car should be developed, trips could be dumped without

uncoupling, and even without stopping, and the bottom layout could be correspondingly simplified.

Neither the cross-over dump nor the bottom dumping car are adapted to installations where the coal is dumped directly from the car into the skip and the "kick-back" dump is too slow for a mine having a large capacity.

The rotary dump combines every advantage of the types mentioned, except the ability to dump a moving trip, which can be done only with the bottom-dumping car. It is capable of dumping with the minimum of breakage and is extremely rapid. It will handle solid-end cars in multiple or without uncoupling. It is rugged, simple, and free from operating troubles. Rotary dumps are used almost universally in England; they are operated by gravity or power or a combination of both.

The best gravity dumper is the multiple compartment type. This is a splendid arrangement for small cars but is not well adapted to a mine-bottom installation because of its large diameter and because it will not handle cars without uncoupling. The Hansen & Hayes gravity dump is rapid but difficult to handle and hard on the coal. The new gravity dumper, controlled by gears and a powerful hand wheel, seems to give perfect control, and for light cars and small capacity should give excellent service.

For large cars and large capacity, power dumpers are almost universally employed. The motor-operated dumpers make a complete revolution; the reversing type, operated by air or steam, revolves through 135° to rest and then returns to its original position. The motor-operated design requires a large motor to handle the peak load in rapid operation. The stop at the end of a revolution is not absolutely positive as it depends on a latch. These dumps are capable of handling eight to ten cars per minute and may be operated by either spur or friction gears from the motor. The reversing type of dump is operated by cable and air cylinders. Speed of operation is obtained with a comparatively low power consumption as the air cylinder takes the peak of the load. A single dumper of this type will handle up to seven cars per minute singly, or five cars per minute in trips. A two-car dumper will handle from six to eight cars in trips and from eight to ten cars in pairs.

The Wood dump is operated by two single-acting air cylinders with tail-rope connection. They are usually set horizontally and with equalized cables running to both ends of the dumping cage. The cylinders are operated by a four-way valve so arranged that one cylinder is filled with high-pressure air from the tank while the other is exhausting into the atmosphere. When the valve is set near the cylinders and is sufficiently delicate in operation, it is possible for a skillful operator to dump rapidly and with perfect control at all times. He can cushion the dump, so that it can hardly be heard when it strikes the stops, or he can

give a blow when necessary to dislodge wet coal or rock. This type of dumper handles the coal easily and without breakage. The cage is provided with a shield on the dumping side, which delivers the coal to the chute without any drop.

The two-car rotary dumper of this type is now made with roller supports at the ends only, thus eliminating the expense of intermediate supports, with the consequent uneven wearing and adjustment of the wheels.

Where the skip contains three or more carloads, and the bottom is laid out according to the system shown in Fig. 5, it is necessary to use a dumper handling the same number of cars as the skip, but where the system shown in Fig. 6 is used, there is no change in arrangement and it is only necessary to provide a weigh pan of sufficient capacity and to arrange the scales for multiple weighing of the requisite number of cars.

Weighing

In most mining fields, the coal is paid for by weight; and even where this system is not in operation, the tendency is to adopt it because it encourages full loading of cars. The coal can be weighed on platform scales or in weigh baskets. The platform scale is the most usual arrangement and, in some respects, the simplest. The principle objection to the platform scale is the variation of the tare weight. Few mines, even new ones, have cars exactly uniform in weight or capacity. In the type of bottom shown in Fig. 5 arranged for a two-car dump, there should be two scales so spaced that two cars can be weighed while two cars are being dumped. In the Kathleen mine of the Union Colliery Co., the trips are handled through the dump, without uncoupling, two scales being used. The scales are on a level grade and cager dogs are set above and below the scale so that the couplings over the scale are slack. This method is thoroughly satisfactory but requires considerable mechanism and a long bottom, which can be avioded if weigh baskets are placed under the rotary dump. The weigh-basket arrangement, shown in Fig. 6, is more elastic, simplifies the handling of cars, and will take care of three or four cars to the skip. The baskets are designed to stop the coal on an easily rounded surface and give an easy discharge without breakage; and while they require one additional handling of the coal, the additional breakage is negligible.

Where two-car rotary dumps are used with a two-car skip, the weigh basket is designed to hold two carloads of coal, each dump being weighed separately by means of a sliding dial scale or tare beam. The dumping does not have to register exactly with the hoisting cycle, as is the case where the other system is employed. The movement of the trip is handled by one man, who also operates the rotary dump. One man

weighs the coal and dumps the weigh pan. The safety gate automatically opens and closes with the skip.

Chutes and Gates

In Figs. 5 and 6, the sump is guarded by a safety gate, which is opened by the skip and is closed by a counterweight when the skip ascends. It is not intended to dump coal against the safety gate when the skip is not in loading position. The gate is ready if a dump should accidentally be made, and can be used, in the weigh-pan design, to hold rock while coal is being dumped into the weigh baskets. The safety gate also catches lumps of coal that are retarded in the chute and thus enables the operator to send the skip away more quickly than would be possible without it.

In operation, the cars or the weigh pans are dumped when the skip is approaching the bottom at such a time as will theoretically bring the coal to the skip as soon as it lands. This is the most rapid operation possible, as the skip is loaded in less time than when the coal starts from rest, as when a measuring hopper is used or when the coal is loaded into the skip direct from a bin.

The safety gate shown in Fig. 5 has a spring catch, with which the skip comes into contact so that the gate floats down from the skip and opens the chute with little impact. A curved lip on the chute makes contact with the skip and prevents all spilling. Fig. 6 shows a design that is better adapted for resisting the impact of coal and rock, which may be dumped against it. This type of gate is operated by a sword-arm lever, which is engaged by the operating roller on the skip. Its motion is conveniently adjustable and the lip projecting from the gate follows the skip down and gives a perfect discharge without any possibility of spill.

A moderate amount of rock can be handled at the skip shaft with either design, provided the hoisting capacity of the plant is proportioned for the rock in addition to the coal, together with a certain proportion of single-car hoists. The loss of hoisting capacity caused by handling rock without auxiliary installation is different in the designs shown. In the case of the design shown in Fig. 5, the worst possible combination of coal and rock is when single cars of rock are interspersed with single cars of coal; in this case, 5 per cent. of rock will then require 10 per cent. additional hoisting capacity, compared with the coal. Where the rock cars in a trip are in pairs, between pairs of coal cars, there will be no loss in hoisting capacity; then a net increase in hoisting of only 5 per cent. is necessary.

In the case of the second design, shown in Fig. 6, the weigh pan would be left open so that the rock would pass down against the safety gate. The worst arrangement of rock and coal would then occur where

two cars of rock are interspersed with two cars of coal, as this would require single hoisting of both rock and coal with the same loss of capacity that would occur in the other case. Where single rock cars are placed between single cars of coal, or the cars of rock are placed altogether in a trip, there will be no loss in hoisting capacity.

In this design, the bottom is level and a reversing power-operated feeder is used for handling the trips. It is therefore possible to consolidate the rock or isolated cars of coal without loss of hoisting capacity, when they are within three cars of each other, by running the cars through the dump without dumping and running them back again so as to select the rock or coal cars and dump them as desired. Where isolated cars of rock occur in the trip, it is necessary to dump them singly, but the reversing feeder avoids the necessity of dumping single cars of coal when the rock and coal cars are not placed in the correct relation in the trip. The dumper can easily take care of such problems, so that the actual number of single-car hoists will be few. The process can be greatly simplified by a little care in making up the trips on the partings.

Where large quantities of rock are to be handled, or where the rock will not flow in a chute, it must be handled at the auxiliary shaft, which is the only way in which a maximum coal-hoisting tonnage can be secured. If an auxiliary rock bin is built under the dump, with mechanical means for feeding the rock to the skip, single hoists of rock will not be necessary but the amount of coal handled will be decreased by the amount of rock hoisted.

The proper handling of rock requires careful investigation in every case, but this problem is perhaps easier of solution in a skip-hoisting plant than in a cage-hoisting plant, because of the greatly increased hoisting capacity of the skips and because there are always two ways of getting out the rock.

Docking

Docking facilities can be arranged by placing a special docking table in the tipple, on to which the contents of a single car may be hoisted and dumped through a fly gate in the chute. This table and the fly gates in the chute should be in charge of the dock boss, who can operate the gate and start and stop the conveyor at will, so that each car of coal sent for inspection may be thoroughly examined and the docks removed and identified. In practice, a single car of coal will be hoisted whenever its condition is suspected, or the dock boss may be given all the cars he can handle in a day's run without unduly shortening the hoisting capacity.

Fig. 7 shows such an arrangement in the tipple together with the rock gate and chute. As the docking table discharges directly onto the feeder or screen, further handling of the coal is unnecessary. The docks can be piled and labeled and afterwards thrown directly into the rock

bin. The operation of the tipple will not be disturbed or delayed in the slightest degree.

Handling the Trips

The foregoing, for the most part, assumes that the cars are handled in trips without uncoupling, which method has proved successful. The labor of uncoupling cars and handling them singly or in pairs is not only expensive, but dangerous; and the ordinary handling of uncoupled cars through a dump or in a cage-hoisting mine, is destructive to the cars and the dump or cage and its mechanism. The only advantages of uncoupling are slightly more elasticity of arrangement and a little greater speed in dumping.

When cars are dumped without uncoupling, it is necessary to use some form of swivel hitching or drawbar. For a reversing rotary dump a three-link hitching properly designed will swivel through the necessary 135°, without sufficient loss in length to cause trouble. Swivel hitchings give good results for small cars and short trips, but when the swivel pin is large, say over 1¼ in. (31.75 mm.) in diameter, the hitching becomes cumbersome and heavy. For large cars, either a swivel drawbar or a three-link hitching must be used. The spring and swivel drawbar head shown in Fig. 8 has given excellent results and combines a spring both for tension and compression with a rugged and inexpensive swivel. The spring drawhead has special advantages where cars are dumped in trips, as the motion of the trip is effectively cushioned both in starting and stopping and the load is handled much more easily by the feeder.

The best arrangement consists of a level bottom, through the cage room and over the dump, with a grade into the empty hole of about 1 per cent. or less. The trips are then propelled through the dump, stopped and held in dumping position by a chain feeder operated by motor and brake, and set in the center of the track as shown in Fig. 6. This feeder is reversible and the trip can be handled, the cars stopped and accurately spotted, by one man with the aid of no other mechanism. The grade of 1 per cent. on the empty side keeps all the couplings below the feeder in tension so that the coupling pins will not fall out in dumping and the cars will be accurately spaced.

Where platform scales or gravity operation is used, cager dogs must be installed above and below the scale, and frequently below the dump. These dogs give slack couplings over the scale, as shown in Fig. 5, and a slack coupling below the dump, where the trips can be cut apart.

Gravity operation of trips without uncoupling should never be attempted except with some kind of mechanical control. Where a two-car dumper is used, the trip must start from rest and run from 18 to 25 ft. into the dogs. A sufficient grade to secure prompt starting and reliable operation will give the cars entirely too much speed when they enter the

dogs on the dump. At the Kathleen mine, see Fig. 5, where gravity operation was used on account of the natural grade of the bottom, a retarding chain feeder was employed, similar to the power-operated feeder except that it was equipped with an air-controlled brake. This brake is applied after the trip gets under way and reduces the speed until the trip lands in the cager dogs without too much impact. Car retarders of the Fairmont type have also been used successfully for this purpose.

A power-operated feeder cannot be safely used with cager horns, unless they are mechanically interlocked. Successful plants have been constructed in this way, but they are quite complicated. Where the platform scales are omitted and the coal is weighed in baskets, the cager dogs are not necessary and the reversing feeder offers the most economical and effective way of handling the trips.

With continuous-trip operation, it is possible to handle the bottom with three men—one to control the trips and operate the dump, one to weigh the coal, and one to pull checks and to cut the trips into the required number of cars below the dump. The switches can be handled by this man or, in a large mine, by a special trip dispatcher. Where the trips are uncoupled, at least two more men are necessary for uncoupling and coupling up the trips on the empty track. In a large shaft mine, even with automatic cagers, at least two or three men are required at the bottom to couple and uncouple the cars; the weighman and check puller are located at the top.

Where the trips are uncoupled and a rotary dumper is used, cager horns must be placed on the dump to hold the cars in dumping position and some means must be provided for preventing the coupling pins from falling out. The impact of the cars against the cager dogs in the dump is destructive to the mechanism of the dump.

Handling Men and Material

In a large mine, an auxiliary hoisting shaft is necessary, especially in a skip installation. This shaft should be equipped with a small tipple, which can be used for handling coal during the development and for hoisting whatever rock cannot be conveniently taken care of in the main shaft. The use of solid-end cars in the rotary dump prevents the use of a self-dumping cage of the ordinary type at this shaft. It was formerly necessary to take the cars off the cage at the tipple landing and dump them singly in a rotary dump, or to dump them at the bottom into an auxiliary skip that had a platform deck for men and material.

In most cases, a single counterweighted cage will be sufficient. This cage should be large enough to handle the mine equipment and to take the men rapidly into the mine. It may have a double deck, the upper deck being designed for cars and the lower deck for men only.

Overturning Self-dumping Cage

The overturning type of self-dumping cage has solved the problem of dumping at the auxiliary tipple. The cage is a simple platform pivoted 3 ft. 4½ in. off center, the motion of the tilting platform being controlled by overhead rollers that run in large radius circle guides. Wheels pivoted on the tipple structure, over which the edge of the cage rolls when the rollers on the cage leave the circle guides, greatly shorten the dumping cycle and retain a large proportion of load on the cable, thus obtaining an easy dumping cycle under perfect control and a quick return of the cage into the shaft.

The dumping cycle is arranged so that the edge of the car projects over the dumping plate just before the angle is reached at which the coal commences to flow from the car. As the dumping cycle progresses, the car turns over above the dumping plate with little drop until the coal is about two-thirds discharged, when the car rises and commences to draw back into the shaft. It was therefore necessary to provide a spill gate; the latest design of this spill gate has a pantograph extension, which catches all the spill from the cage platform as well.

The spill gate is operated by a lever on the tipple, which is engaged by angle guides on the bale of the cage. The gate is drawn back and nested under the dumping chute, by means of its own weight aided by the counterweight.

The cars are retained on the cage by keepers similar to those employed in an ordinary self-dumping cage. Where brakes are not used, there is a shelf engagement over the wheels, in addition to dogs that engage the wheels or lugs on the car. The latest designs of this cage have keepers of the Lepley type, which go partly around both wheels and hold them against sliding and overturning.

Ventilation

Dumping at the bottom of a downcast main shaft is sometimes objected to on the grounds that the dust produced may be carried into the main haulage ways. This objection may be removed by making the hoist compartment of the auxiliary shaft the downcast and the air compartment the upcast, with the main shaft on practically dead air. The opening of the dump to the shaft can be entirely sealed off, except for the chutes which are open only when the skip is loaded. It may also be possible or desirable to provide a separate air shaft, perhaps of circular form, in which case the auxiliary hoisting shaft will be downcast and the main shaft, as before, will be on dead air.

Cage Room on Mine Bottom

In a skip installation, the cage room and loading tracks run parallel with the long way of the shaft whereas in a cage-hoisting mine the loading tracks are perpendicular to this plane.

It is desirable in arranging the mine bottom for either skip or cage hoist to locate the auxiliary shaft so that cars may be readily detached from trips and sent to the auxiliary hoist without serious interruption to main operations. This auxiliary shaft should also be located with full consideration for getting cars, mine machines, material, and men to and from the main workings. The mine motor pits and repair shop, if any, are located where the haulage motors, crippled machines, and cars may be readily shunted from the main haulage roads and also convenient to the auxiliary hoisting shaft.

Numerous variations are possible in the arrangement of entries, just as in cage hoisting, but it is necessary to place the load tracks parallel with the long way of the shaft. Numerous efforts have been made to adapt the revolving dump and skips to an existing cage hoist layout, but to the writers' knowledge no entirely satisfactory method has as yet been worked out.

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DISCUSSION

R. V. Norris, Wilkes-Barre, Pa.—The question of breakage should be studied. I am not prepared to accept offhand the proposition that there is none, as I have found considerable breakage in anthracite under similar conditions, but I do not believe that the breakage is sufficient to offset the advantages of the skip hoisting, where large capacity is needed.

Geo. S. Rice,* Washington, D. C.—In Alabama, the Tennessee Coal and Iron Co. early used skips. It used an interesting type of revolving dump in the hopper, in which three cars were used, so that two were upset to carry over. Where you have one kind of hoist, and not too much rock, skip hoisting is the right thing, but where you are hoisting different kinds of coal, as in Europe where they mine eight or ten seams in one shaft, it is necessary to use cars.

I am glad to see the tendency toward the use of tight cars in coal mines, because it is a great protection in the matter of dust. Close observation and experimental tests have indicated that there is a certain advantage in the use of tight cars. I should not think, with the supplemental hoist Mr. Allen mentions, that it would be necessary to provide for open-end cars.

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* Chief Mining Engineer, U. S. Bureau of Mines.

H. N. Eavenson, Pittsburgh, Pa.—About two months ago, it was my privilege to see the plants in Illinois, to which Mr. Allen referred; also the plant that now holds the record for self-dumping cage hoisting. None of the plants was working to capacity but there was a decided difference in the attitude of the men around the shaft bottoms. Where they were dumping from cages, the hoist was 680 ft. and the record is over 1530 cars, I believe, in 8 hr., or an average of slightly over 3.1 hoists per minute, for the entire working time. The day we were there, they were running at the rate of 2.7 hoists a minute. When we began to time the hoisting, the men immediately began to speed up. But at the skip plant, nobody paid any special attention to us, nor to the output of the mine, which was comparatively new. I timed the hoist and found that the cycle was running 48 to 50 sec. and they were hoisting 650 tons an hour. As the cycle to which the plant was designed was 32 sec., the capacity could be almost doubled from what it was then.

Figures on breakage are hard to get, but the owners of the plant that holds the record for hoisting said they had made a careful comparison of the figures in one of the skip plants with those for their own mine, going back eight or ten years or more, and had decided that there was about 2 per cent. less breakage with skip hoisting than with the cage. They are going to install skips in a proposed new plant.

My opinion is that the breakage is little, if any, more with skip hoisting than with cage hoisting. At the cage-hoisting shaft, every day from 8 to 10 tons of coal was gathered from the ground landing, and from 20 to 30 tons from the bottom of the shaft. In fact, this was such an item that the owners were planning to install a bin under the cage landing, from which the spillage could be removed in mine cars. It was costing nearly $30 a day to clean up this coal. In the case of shallow shafts, the power consumption for skip hoisting would be less, in proportion, than for the self-dumping cage.

E. A. Holbrook,* Washington, D. C.—In 1902, the Dominion Coal Co., at its No. 2 colliery, at Glace Bay, Nova Scotia, having mined the Harbor seam of coal 400 ft. below the surface, decided to sink the shaft to the Phalen seam, 850 ft. below the surface. It was desired to have a hoisting capacity of 6000 tons per day from the lower seam; therefore the company decided to install skips, or hoisting tanks, each holding 6 tons of coal. In 1911, it was hoisting over 4000 tons a day. The officials praised highly the working of the installation, low cost of upkeep, and the fact that in the mine, which was very dusty, tight-end cars could be used.

Robert Obert.—The southern anthracite region has heavy pitching veins, from which the coal cannot be mined out as in the soft coal or

* Assistant Director, U. S. Bureau of Mines.

the upper anthracite region, where it is mostly hand loading. All of the coal is loaded from chutes and in going through the mine for the second or third mining, it is necessary to use a great deal of timber. As this is loaded out in the car, it will get in the hopper and under the rotary dump, which will interfere with any of the cut-off jigs on any of the intermediate levels.

ANDREWS ALLEN.—As far as hoisting from different levels is concerned, the metal mining practice may be used there, but the loading gates cannot be handled in quite the same way. I do not believe the problem is insoluble but I cannot say offhand how it ought to be handled.

T. M. CHANCE, Philadelphia, Pa.—One of the most important elements in the success of skip hoisting, as applied to shaft mines that are to be intensively operated, is the loading arrangement used for charging the empty skip. Some years ago I collected data on this subject. The toggle-operated drop-bottom loading pocket, used at the Newport mine on the Gogebic Range, seemed to be one of the best devices then in use. I believe this type of gate was first used at the Chapin mine in the Lake Superior district, although I understand that the idea originated in South Africa. At the same time Mr. Catlin installed, at Franklin Furnace, a type of undercut loading gate for the charging of the loading pocket from the storage bin; this method of loading the pocket prior to its being emptied into the skip appeared to be entirely satisfactory for the loading of ore that breaks into large masses and which might contain large pieces of mine timber. The Newport pockets were fitted with overcut gates for the charging of the skip pocket, and while these seemed to give entire satisfaction when used on the soft ore of the Gogebic Range, they are not so well suited to the handling of ore that does not readily break into small pieces as the undercut gates used at Franklin Furnace. A combination of the Newport method of charging the skip and the Franklin Furnace method of loading the pocket should work admirably.

In considering the possibility of skip hoisting in the southern anthracite regions, the installation of loading devices capable of handling large rock masses together with large pieces of mine timber is of great importance. In working the steep pitching breasts common to this field, everything that is broken must be drawn and loaded into the mine car.

Large pieces of roof rock frequently are drawn into the chutes and these are difficult to pass through the common forms of rock gates. While undercut gates have been used in some of the anthracite collieries, I do not recall their use under conditions in which the material running out of the chute flows to its angle of repose, as at Franklin Furnace. Under this latter method of operation large pieces of rock can be bulldozed in the chute and large pieces of timber can be readily removed. If possible to avoid it, no part of skip-loading devices should project into the shaft.

This may not prove feasible in anthracite mines, because if breakage is to be minimized the coal should slide out of the loading chute into the loading pocket and from the loading pocket into the skip. To attain this end, the skip may be run into a recess in the wall so that the body, when loading, lies at an angle to the vertical. The loading chutes might then be designed to project into the skip during loading so that the coal in the loading pocket will gently slide from that pocket across the loading chute into the skip. Then the drop of the coal would be greatly reduced, as well as the breakage that might be caused by dropping the coal into a vertical measuring pocket prior to loading it into the skip. Breakage would further be reduced if the loading pocket were a slightly inclined chute rather than a deep bin, as crowding of the coal through the discharge opening would be avoided; and, second, there would be a decrease of weight on the hoisting rope at the time of accelerating the load. With the present system, when the loaded skip goes into the headframe dump the weight of the skip is reduced by the amount that is supported in the headframe. This is admirably shown in the moment diagrams in the paper. At the beginning of the wind it would be desirable to have the full weight of the empty skip in the headframe upon the rope, as then the moment due to the loaded skip and rope in the shaft is the greatest. If, however, the loaded skip is resting in a recess in the side of the shaft, the moment due to its weight and the weight of the coal that it contains is lessened and the resultant stress existing during the early portion of acceleration is correspondingly reduced. I believe that a method to reduce the moment during acceleration in this manner has been employed at the Vulcan mine, although in this case a single skip was hoisted in balance with a counter-weight, which runs off at an angle at the end of the wind so that its moment is reduced during acceleration.

Mr. Rice has referred to the difficulties attendant on the use of skip hoisting in multiple-level mines, such as those in the southern anthracite region. At the Dominion Iron and Steel Co., this problem of multiple-level skip hoisting was solved by the use of two compartments of the shaft for raising the coal from the lower seam and two compartments for coal from the upper seam. One of the difficulties in multiple-level skip hoisting is the spill that occurs during loading; it would seem that this method of the Dominion Iron and Steel Co. would go far to solve this phase of the problem, as each portion of the shaft can be completely isolated.

In the last few years motor-driven hoists of large capacity have been installed. While at first the induction-motor hoist would appear far superior from the standpoint of first cost, simplicity, etc., some of the larger mines that have been fitted with such hoists might more profitably have been equipped with direct-current hoists, operating with controls of the Ward-Leonard type. This, however, could not have been done in

most of these cases unless the duty cycle of the hoisting operation were changed to fit the electrical requirements of the hoist.

One essential to proper hoisting is the elimination of acceleration peaks; these can best be avoided by hoisting continuously with minimum time for loading and with minimum deadweight of the device used for holding the material to be hoisted. The skip hoist fulfills both of these requirements in that the weight of skip can be made considerably less than that of a self-dumping cage and mine cars, and the time for loading unit capacity can generally be appreciably reduced below that required for caging the same capacity in mine cars.

This point is well shown in two hoists designed at the same time for almost the same duty. Both hoists were to operate to about the same maximum depth, 2400 ft., and from about the same center of gravity in the mine, 1200 ft. Hoist A was to raise material in skips of high capacity whereas hoist B was equipped for raising the cage and mine cars then in use at the mine The maximum rope speed of hoist A was 1100 ft. per min., that of hoist B 3500 ft. per min. Due to the greater rope speed, the length of time for caging cars, and the greater deadweight necessary to be hoisted to raise the same amount of material in 24 hr., the horsepower during acceleration with hoist B was about 1500, whereas with hoist A, operating with a minimum time for loading and unloading and a minimum deadweight of skip, the accelerating peak was about 750 hp. In both cases the average horsepower during the wind was about 525. We thus have two hoists of practically identical duty, fitted with Ward-Leonard control, and therefore with the same general electrical devices on the hoist itself, but one, hoist B, required a flywheel motor-generator of the Ilgner type, whereas the other, hoist A, was fitted with a motor-generator set equipped with a synchronous motor allowing operation with leading power factor. Hoist B is a detriment to the electrical system to which it is connected, owing to the slip control employed with the Ilgner set and will probably require concessions to be made in the power contract due to its disturbance of line power factor, whereas hoist A is an improvement to the entire electrical distribution system and should therefore be allowed a considerable reduction in unit power charges, due to the condenser effect of its motor-generator set.

Andrews Allen (author's reply to discussion).—Mr. Eugene McAulife, president of the Union Colliery Co., which operates a skip-hoisting plant at the Kathleen mine, DuQuoin, Ill., recently said that the chief characteristic of a skip plant is "tonnage without hysteria." This is what Mr. Eavenson noticed and is typical of a plant of this type. There is no reason, however, why a skip cannot be hoisted as fast or faster than a cage. There is less hazard and a much smoother operation. The objections to speed in a skip are only that it is needless and expensive.

There may be cases, however, especially in connection with old installations where the shaft is small and in deep operation where a large shaft is expensive, where it may be economical to hoist much faster than the speeds mentioned in the text.

In closing the discussion, it may be well to note that some interesting ideas in mine-bottom arrangement for handling continuous trips from the rotary dump have been worked out since the paper was read. It has been the writer's aim to eliminate wide entries as far as possible and the latest designs embody single-track entries, both for empties and loads. The empties are handled by a locomotive, which pushes the empty trip up either run-around to a point where the haulage motor can pick it up with the minimum loss of time. In this design there is no possibility of interference or delay and one man and a motor will do all the work on the empty side, even in the largest mine. On the load side, there are a number of track chain feeders spaced at intervals a little less than the length of a trip. These feeders are electrically connected and operated by the dumper, who is thus enabled to bring down the trips as he wants them and couple them on to the cars standing on the main feeder. In this case the haulage locomotive leaves its trip over one of the several track feeders, then runs through a break-through, picks up its empties and goes back into the mine.

It may be interesting to note that what seems to be a satisfactory and reliable bottom dumping car is now on the market. This means that the advantages of this type of equipment are now available in connection with skip hoisting in cases where a deep pocket with consequent breakage is unimportant, and especially when the number of cars is relatively small. This car so far has met every test and seems to overcome all the mechanical objections enumerated in the paper.

The possibilities of a skip installation are so numerous and its applications so elastic that there seems to be no limit to its application. A recent analysis showed important economies as against a slope installation for a 6000-ton mine at a depth of only 115 ft. (35 m.). Another 1500-ton operation with a single counterweighted skip showed decided advantages over any other type of installation for a depth of 200 ft. It would be interesting to determine for different depths and tonnages the most economical size, hoisting cycle, and arrangement of a skip-hoisting plant. So far, this has not been done, for the reason that the advantages of the system itself are so many that the size of units has been dictated by convenience rather than by the last percentage of hoisting economy. The newest and most up-to-date plant in Franklin County, Ill., is now being designed for the Chicago, Wilmington and Franklin Coal Co., with 13-ton skips and a maximum output of 1250 tons per hour.

Coal-pillar Drawing Methods in Europe*

BY GEORGE S. RICE,† WASHINGTON, D. C.

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SOME form of longwall mining is generally used in Continental Europe; also in Great Britain where the coal is weak and friable, or the coal bed provides material for pack walls and filling, or where the bottom is soft and squeezes up easily, or the roof is pliable, or the bed is thin and brushing provides building material, or the thick multiple seams are mined in layers, such as the 24-ft. seam at Weymss, Scotland, or the 10-yd. pitching bed near Coventry, England. Pillar methods have been retained in British fields where the coal beds are from 5 to 9 ft. (1.5 to 2.7 m.) thick, and free from thick partings or binders or without draw slate, which would provide waste rock for pack walls, or where the roof is hard and requires shooting to bring down, and also the bottom or floor is comparatively hard.

The room-and-pillar system, by which probably 95 per cent. of the coal of the United States is produced, is now found in only a few mines in Wales, where it is known as the pillar and (single) stall method. The American room-and-pillar system is not equivalent to the bord-and-pillar or stoop-and-room system. The essential difference between the bord-and-pillar and the average American room-and-pillar system is probably due to natural conditions. The majority of the coal mines in the United States have a depth of less than 700 ft. (213 m.). The bulk of the coal mined in Europe comes from a depth of more than 1200 ft., while the deepest mines in Great Britain, Belgium, and France reach 3500 to 4000 ft. In deep workings wide spans are impracticable, even with the use of abundant timber; accordingly, as the mines in the United States become deeper the rooms will become narrower and the pillars thicker, and the bulk of the coal will be extracted on the retreat.

It is generally conceded, without reference to the cost of production, that the larger the pillars left on the first mining, the more thoroughly can the coal be extracted. In Europe complete extraction is generally compelled either by the lessors or by the governments; whereas in this country it has been more largely a question of competitive cost of production, or the support of the surface in the flat farming districts of the

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* Presented by permission of the Director of the U. S. Bureau of Mines.

† Chief Mining Engineer, U. S. Bureau of Mines.

Middle West, that has determined how completely the coal is to be extracted.

Generally, the more complete the extraction of a coal deposit, beyond 70 to 80 per cent., the higher will be the cost of production, due to the necessity for more extensive timbering or the building of packs and walls. Although by the longwall system practically all of the coal is excavated, the cost of production is greater than by the room-and-pillar system, because of the large amount of dead work. On the other hand, with heavy overburden accompanied usually by friable coal and roof, it is often the only system that can be used.

The earlier wasteful methods in room-and-pillar mining in the United States have been difficult to overcome, partly because the miners' contract prices have largely been determined on the basis of wide working places, wide crosscuts between rooms, and narrow pillars. These agreements, with the customs and usages that have been established, are hard to change when the mining proceeds under heavier cover or more complete recovery is demanded by the owners of the coal fee.

Pillar-mining Systems in Use in Europe

Systems employing pillars, used in Europe, may be classified as follows: Pillar and (single) stall; pillar and double stall; square chamber method of South Staffordshire; bord and pillar, or stoop and room; rooms or bords hydraulically sand filled.

Pillar and Single Stall

Pillar and single stall, the equivalent of the American room and pillar, is used in a few places in Wales. It was observed by the writer in the Craig Merthyr colliery, in a slightly pitching and shallow anthracite bed 5½ ft. (1.65 m.) thick with strong roof. The stalls were 9 yd. wide and the pillars 6 yd. thick. The pillars were extracted by the methods generally used in American mines, starting at the inner end of the pillar and bringing back a block of pillars simultaneously.

In Upper Silesia, room-and-pillar methods of ordinary type were used extensively in beds 10 to 60 ft. (3 to 18 m.) thick; but owing to extensive fires and trouble from subsidence of surface and generally poor recovery of coal, these methods have been supplanted by longwall and hydraulic sand-filling methods.

Pillar and Double Stall

This method was formerly used in Scotland and, in 1911, was used in Wales at the Seven Sisters colliery in an anthracite bed 7 ft. to 8½ ft. (2 to 2.5 m.) thick. The stalls are 42 to 48 ft. wide and the pillars 36 to

42 ft. wide. In this system, as shown in Fig. 1, the stalls have two entrances and the tracks are at either rib, the excavation between the tracks

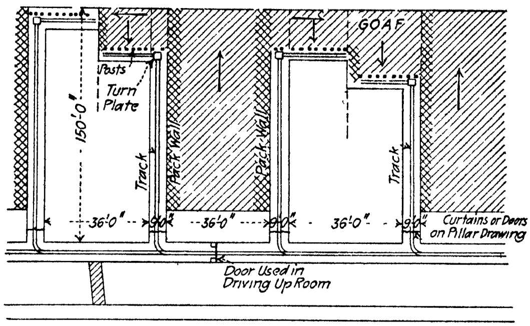

FIG. 1.—PILLAR-AND-STALL METHOD.

being gobbed. As soon as the stall has reached the next entry or level, it is widened on either side and slices about 10 to 18 ft. wide are taken back; sometimes a thin pillar is left between stalls, but this is lost. The average recovery is claimed to be about 90 per cent. Gradually the system is being supplanted by longwall. This system was formerly used in Appanoose County, Iowa, in a 30 to 36-in. bed, but the thin pillars were not generally recovered; the system has been largely replaced by longwall.

Square-chamber Method of South Staffordshire

This unique system, shown in Fig. 2, was developed to mine the so-called Ten Yard seam which lies approximately horizontal. The cham-

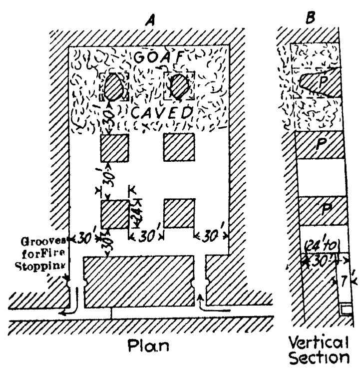

FIG. 2.—SQUARE-CHAMBER METHOD OF SOUTH STAFFORDSHIRE.

bers, which are separated by thick pillars, are 46 yd. (42 m.) wide and up to 200 ft. long, arranged with four to six internal coal pillars. They are

opened with two narrow gate roads off the level. As the coal is liable to fire spontaneously, arrangements are usually made for fire dams, grooves being cut in each rib of the gateway and clay being placed conveniently at hand. These gate roads are connected by a "lane" 10 yd. (9 m.) wide; also stalls or rooms 10 yd. wide are driven at 8-yd. intervals extending the gates, and at every 8-yd. interval there are similar cross lanes 10 yd. wide, thus leaving internal coal pillars 8 yd. square.

This mining is in the bottom coal and to a height of 7 to 8 ft. (2 to 2.4 m.). After the chamber is formed, the top coal is taken down in sections, slice after slice being removed vertically, by cutting vertical grooves successively every 6 ft., leaving "spurns," or narrow webs holed through, in the upper part. When the layer of coal that is being extracted is thus cut, the spurns are reduced by pick and finally knocked out with a "picker," which is like a boathook. It is so managed that the mass falls as a whole and the coal is then loaded. Thus the work continues until the roof is reached, when the internal pillars are sliced away as much as possible before the roof falls. Necessarily more or less coal is buried by the falls. The thick pillars between chambers are extracted after the ground has settled.

No appreciable amount of methane has been found during the past hundred years of working, although the bed reaches a depth of from 1200 to 1500 ft. (365 to 457 m.). The mine is worked with open lights and sometimes with candles. The chambers and roadways are dusty but it is said that no explosions have occurred in the field except during the war, when one resulted from blasting. The infrequency of the explosions, apart from the fact that the coal is not of a very inflammable character, is probably due to the small use of explosives, the coal mainly being extracted by pick and bar.

An unique feature in the South Staffordshire field is the "bumps," which apparently are due to the strong overlying rocks that span underground subsidences due to mining and which, when they break, produce a jar that throws down the immediate roof and smashes timbers. Generally there is some preliminary dribbling from the roof, which warns the miners, so that comparatively few serious accidents have occurred from this cause. A similar condition prevailed in a mine at Fernie in British Columbia, but there the "bumps" caused considerable loss of life. In that mine, the condition has apparently been overcome by taking out only a small percentage of the coal by entries that divide the coal area into large pillars, as in the bord-and-pillar system, the bulk of the coal will then be recovered by what is equivalent to longwall retreating.

The yield of coal by first mining in the South Staffordshire field seems to be low, and in spite of a large number of remaining pillars considerable coal is lost by the caving of the roof. It requires skilled workmen and does not appear to have much to recommend it for use elsewhere.

Bord and Pillar or Stoop and Room

In Europe, this is the most important system in which pillars are used. It is common in the north of England, in Scotland, and to a lesser extent in Wales; in a modified form it is used in the Upper Silesian field in connection with the hydraulic method; see Figs. 3, 4, and 5.

In the typical bord-and-pillar system narrow places are driven at right angles to one another, forming large square or rectangular pillars that are extracted when the workings reach the boundary. With the best conditions, this is essentially longwall retreating and is then an admirable system. Little or no explosive is required, as the coal "facings" are usually close together in European coal beds and pick work takes the place of explosives.

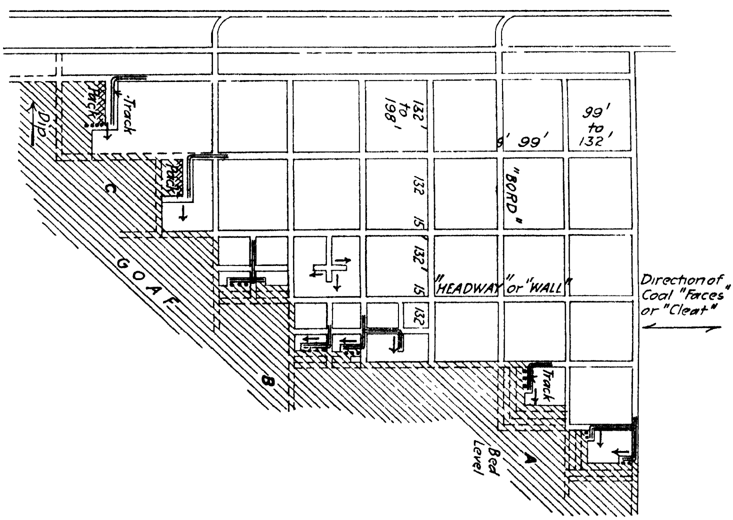

FIG. 3.—BORD-AND-PILLAR METHOD.

Formerly this system was used almost universally in Great Britain in the flatter beds, but gradually the longwall supplanted it until now probably over three-fourths of the coal is produced by advancing longwall. It is found in the same districts as the advancing longwall and sometimes in different beds of the same mine. It is considered especially applicable, in fact necessary, where the coal bed is approximately horizontal or not dipping over 1 in 3, has a medium thickness (4 to 10 ft.), is free from thick partings, and the roof is strong and not flexible or bending in character. Conversely, longwall is used where the roof is weak or flexible, or where the coal bed is comparatively thin (under 4 ft. thick) or where the coal bed is thick and there is a soft underlying clay, or where the bed has thick partings that can be used for pack walls.

While the bord-and-pillar practice has not been standardized, it is probable that the deeper the workings, the more necessary it is to have large pillars compared to the width of the surrounding places. The essential feature of the system is to drive "headways," or what would be termed butt entrirs in the United States, about 3 yd. (2.7 m.) wide advancing on the ends or butts of the coal, and from these are turned off bords, or rooms, 3 to 5 yd. wide advancing on or at right angles to the facings of the coal.

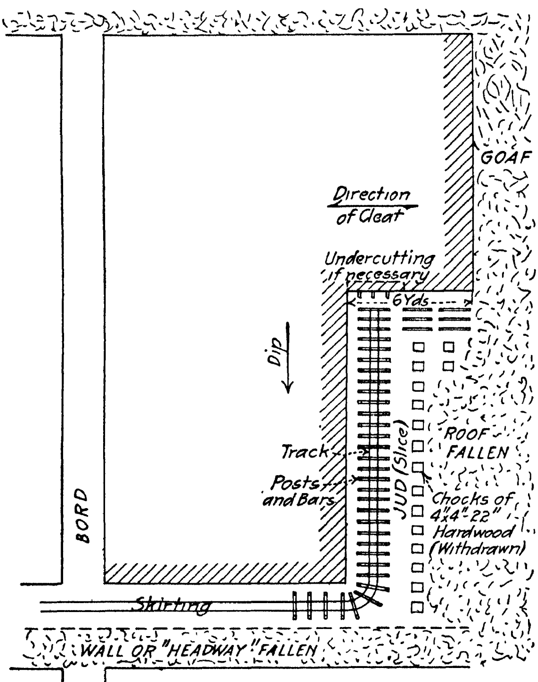

FIG. 4.—BORD-AND-PILLAR METHOD. DETAILS OF PILLAR EXTRACTION.

A typical case is at the Eppleton colliery near Durham, England where the mines operate three coal beds which lie practically horizontal at depths of 1008, 1050, and 1170 ft. (307, 320 and 357 m.) and have thicknesses of 7 ft. 4 in., 6 ft. 10 in., and 3 ft. 8 in., respectively. The pillars vary from 33 by 44 yd. to 66 yd. square. The headways or walls (driven on the butts) are 3 yd. wide and the bords, which work at right angles to the coal facings, are 5 yd. wide. There is therefore extracted, in forming the pillars, only 10 to 17 per cent. of the coal. The final recovery on the retreat is good, and it is claimed that the total recovery is 95 per cent.

The pillars are extracted in various ways but, in general, when the boundaries are reached, the pillars are brought back on a stepped diagonal line. The pillars to be drawn are split by a headway and a narrow bord, making four smaller pillars, then these smaller pillars are split or successively sliced off either on the goaf side or, where the roof is of a character that breaks short, by slicing off from the sides of the headings working toward the goaf. Where the roof is poor, it is sometimes difficult to make complete recovery and small corner stumps left for protection are lost.

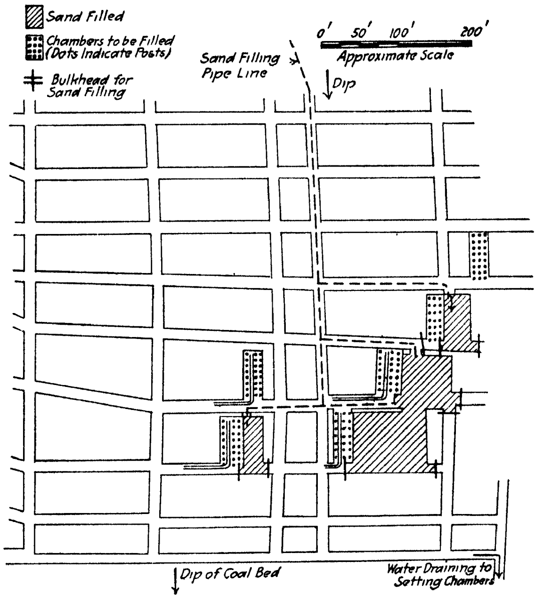

FIG. 5.—BORD-AND-PILLAR METHOD USED IN UPPER SILESIAN FIELDS. PORTION OF DELBRÜCK MINE.

The system, in general, is admirable, the recovery of the coal being only slightly inferior to longwall advancing in that respect. It is not, however, well adapted to the use of mining machines and no instances were brought to the attention of the writer where machines were being used. In the withdrawal of individual pillars, slices are taken off the goaf side; lines of breaker props are sometimes used and withdrawn, in other places cogs are employed. In general, it is expected that the timber will be almost completely recovered. In some instances when the pillars and the workings are old or the roof or floor is not strong, the headways and bords become filled, in which case it is generally cheaper to drive places alongside of them. These places are called "jenkins" and "skirtings," respectively, in the Durham district.

Objections to the bord-and-pillar system are the large amount of narrow work at the start, resulting in the production of a large proportion of small coal or screenings from such narrow places and the payment of yardage to the miners. These conditions are sometimes obviated by making both the headways and the bords 5 yd. wide and allowing the roof to cave in, except such passages as are needed for haulage or air ways.

Rooms or Bords Hydraulically Sand Filled

This system was found only in Upper Silesia, where it is used in the flatter coal beds, up to thicknesses of 10 or 12 ft. In the thicker multiple beds (up to 60 ft. in thickness in the Myslowitz Colliery), the longwall slicing system is employed, in some instances layer by layer, and in steeply pitching beds (as in the Delbrück Colliery) successive horizontal slicing followed by sand filling is used. The bord workings are driven like narrow rooms from the level or gangway, forming large rectangular pillars, which are extracted by successive slices from gangway to gangway. As fast as the slices are extracted the spaces are filled hydraulically with sand and gravel by the method introduced from the anthracite district of Pennsylvania in 1897 and developed to meet the particular conditions. The recovery is apparently very complete. This method has succeeded in preventing the spontaneous fires, which were a serious menace in mining these thick coal beds by the room or chamber method. The only serious objection is the loss of timber, as little of the timber is recovered from the props and lagging necessary to support the roof while the mining is going on. In Upper Silesia, extensive sand and gravel beds are adjacent to or in the neighborhood of the mines. Under pre-war conditions, the sand filling added only about 25 c. per ton to the cost of the coal, which was more than repaid by the freedom from fires and from explosions of coal dust, and by the nearly complete extraction.

Summary

1. In Europe, only a small proportion of the coal production is obtained by mining systems employing pillars other than shaft pillars, pillars under buildings, or as barriers.
2. Pillar systems are employed to a limited extent in Great Britain and in connection with sand filling in Upper Silesia.
3. The typical American room-and-pillar system is not employed in Europe except in a few places in Wales.
4. The principal pillar system, where it is used at all, is the bord-and-pillar, in which the pillars are extensive, compared with the area taken out by the preliminary bords and headways; in general not over 10 or 15 per cent. of the coal is taken out in advancing.
5. In its best form, the bord-and-pillar system permits a recovery of

95 per cent. of the coal, by what is practically a retreating longwall method.

6. The pillars are extracted by successive splitting or slicing.

7. The bord-and-pillar system is applicable only where the coal and roof are relatively strong and the coal is free from large partings, and there is no draw slate which requires gobbing.

8. The American room-and-pillar system approaches the bord-and-pillar where the rooms are narrow and the pillars wide and the retreat is carried on a diagonal line, as in mines of the Connellsville district of Pennsylvania, and in some of the deep mines in the Rocky Mountain region.

DISCUSSION

Carl Scholz, Charleston, W. Va. (written discussion).—This paper is timely, because greater interest is being taken in the more complete extraction of coal in the United States on account of the higher land values and the more stringent requirements of the lessors.

Fig. 6 shows the plan adopted for the Valier Coal Co., at Valier, Ill., where the No. 6 seam, with a thickness of from 7½ to 12 ft., lies at a depth of 650 ft. The roof over this coal is a very soft shale which slacks readily when it comes into contact with air. Therefore the workings are driven so as to leave anywhere from 1 to 5 ft. of top coal for roof protection.

The rooms, 25 ft. wide, are driven on 85-ft. centers, leaving 60-ft. pillars between the rooms, and as the mining law requires a breakthrough every 60 ft., the pillars remaining are 60 ft. square. Between every two panels, 25-ft. pillars are left so that each panel can be completely worked out and remain isolated from the other workings.

The main entries are driven on the four-entry system, the butt entries on the three-entry system, and the panel entries on the two-entry system. Pillars 175 ft. wide protect the entries so that the panels can be entirely worked out without fear of carrying any squeeze over on to the entries. The exact manner in which the pillars will be drawn has not yet been determined; but it is believed that the best results will be obtained by slabbing the pillars along the face on the retreating plan and that it will be possible to recover the top coal over the rooms and extract the pillars completely. Inasmuch as the overlying measures are principally shale and soft slate, it is anticipated that the roof will cave readily as the pillars are extracted. Quite likely cribs will have to be employed, and on account of the great amount of coal remaining in the roof and pillars, considerable expense will be justified in this protection. In order to avoid too large openings, breakthroughs between rooms are staggered.

It is obvious that by this method a larger extraction can be obtained than is the case in Franklin County at this time. The customary method is to drive rooms on 40-ft. centers 24 ft. wide, leaving 16-ft. pillars. The extraction, as determined by C. M. Young, of the University of Illinois, averages 41.4 per cent.[1] It is believed that, by the scheme adopted at Valier, the extraction will be at least 70 per cent. The disadvantages of this scheme are that, on account of their length, tracks must be laid in each breakthrough, but as these breakthroughs are 20

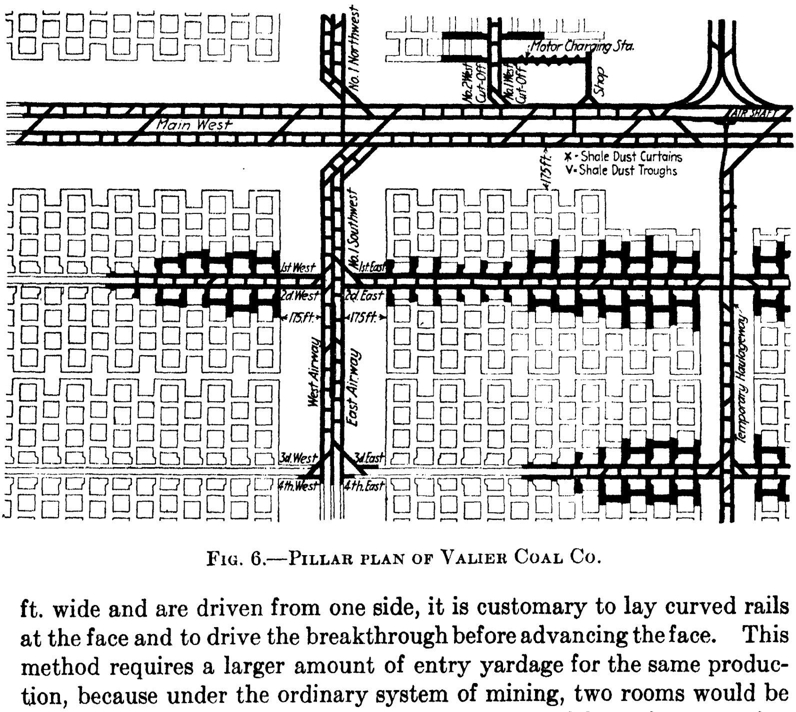

FIG. 6.—PILLAR PLAN OF VALIER COAL CO.

ft. wide and are driven from one side, it is customary to lay curved rails at the face and to drive the breakthrough before advancing the face. This method requires a larger amount of entry yardage for the same production, because under the ordinary system of mining, two rooms would be turned against only one at Valier. As a matter of fact, the extraction of coal is almost as great as in the ordinary scheme, because the breakthroughs take out nearly as much coal as would ordinarily be obtained by two rooms running parallel to each other. The main object of this scheme was to make the pillars square rather than long and narrow, because this shape affords better support and resists pressure more readily.

D. HARRINGTON,* Denver, Colo. (written discussion).—It appears that, probably due to overlying weight from depth of workings, compara-

[1] Univ. of Illinois *Bull.* 100, 47.

* Supervising Mining Engineer, U. S. Bureau of Mines.

tively little explosive is used in European mines and much of the coal is extracted by pick work alone. Presumably the coal thus obtained comes in comparatively finely divided form; it would be interesting to know whether the proportion of lump coal obtained is not much less than the American market demands.

It would also appear that mining machines are not used in Europe as extensively as in America. If this is so, is it due to excessive pressure on the coal or to some restrictions by labor or to former customs, etc.? Are the types of machines in use superior to ours?

Though our popular room-and-pillar system is infrequently used in Europe, the systems in vogue in the seams 5 to 10 ft. thick (in general much more expensive systems than ours) do not give a much better percentage of total seam extraction, while the systems used in the thick seams (above 10 ft. thick) are almost as wasteful as ours, except that the sand-filling methods of Germany are apparently effective.

As to the sand filling in Upper Silesia, Delbruck Colliery, I do not quite understand how the sand filling in a slice can be kept separated from the coal afterward mined out of an adjoining slice, especially as mining one slice and filling an adjoining one seem to be practically simultaneous operations, or, at best, the performance of one operation quickly succeeds the completion of the other.

What is the maximum thickness of coal seams in which Europeans apply longwall methods? Also to what limit of pitch is longwall work applicable?

Has Europe designed any method of mining whereby more than 75 per cent. of thick seams (say 12 ft. or over) can be extracted? In mining thick seams, generally coal is left as roof in advancing workings; and in retreating, while one may get most of the pillars, the top coal must be largely lost, or if the top coal is recovered, the pillar is largely lost; has Europe been able to avoid this loss? What European practices can be adopted to advantage in the coal mines of the United States and what practices should we avoid?

H. I. Smith, Denver, Colo. (written discussion).—In southern Europe in the kingdom of the Serbs, Croats and Slovenes, made up of Serbia, Macedonia, and the Slavic provinces of the former Austrian and Hungarian kingdom, are many coal fields of semi-bituminous and lignite type with a smaller amount of higher-grade non-coking coals. The beds worked range from about 2 ft. (0.6 m.) to 165 ft. (50 m.) and lie from practically horizontal to vertical. Some beds are very regular in thickness and quality, some have no partings, others have partings varying in thickness up to one-third of the total thickness; other beds are folded and faulted and still others are lenticular deposits. In 1919, 2,506,000 tons of coal were produced. This production of coal, including the Pecuj coal mines (which were awarded to Hungary but the coal mined was

awarded to Jugoslavia) was, with the number of mines in each province, as follows:

Province	Number of Mines 1920	Tons, Metric, 1919
Slovenia	21	1,152,000
Bosnia	8	679,000
Pecuj (Hungary)	7	311,000
Croatia	30	260,000
Serbia	18	104,000
	84	2,506,000

Practically all of these mines were visited by the writer in 1920. The mines operating in 1919 were practically identical with the 84 operating in 1920. The amount of coal produced in 1920 was probably close to 3,000,000 tons; the capacity of the mines was about 1,000,000 tons more, depending on better organization of the railroad forces and additional railroad equipment. All the mines in Bosnia were government owned, as were a few mines in the other provinces. The men in charge, in practically all cases, were educated in central Europe, so that central European methods were closely followed.

The lack of efficient mining was greatly in evidence, especially as regards the extraction of pillars. A large quantity of pillar coal was lost by fires, poor ventilation, insufficient prospecting of the bed in advance of mining, the lack of accurate mining records; there was no systematic checking by the higher officials. Fires from spontaneous combustion made many areas inaccessible for mining operations and caused a corresponding loss of the pillar coal, not only in the sealed area but in the adjacent workings. The ventilation with few exceptions was deficient, making it difficult to reclaim pillars adjacent to the fire areas, which would have been feasible with good ventilation. The poor ventilation was no doubt largely responsible for the fires.

Accurate maps showing the coal actually extracted and that lost in pillars were not kept; invariably an abandoned section was indicated by hatching, indicating complete extraction. The tonnage derived from the mine or sections of the mine were not checked against the amount of coal originally in the mine, or sections, so as to determine the actual efficiency in recovering the available coal. In the thicker beds, much coal was lost in the roof. In other sections, large quantities of coal were lost due to the thickening of the partings; in some cases but half the available coal was mined and in other cases the entire area was abandoned. None of these things were shown on the maps, which indicated clear extraction. Where hydraulic filling was used, the extraction was much greater than where filling was not used.

The bord-and-pillar method of mining was the more popular, though the dimensions varied. A projected plan worked out for the Bogovina mine in Serbia is shown in Fig. 7. This bed of coal was from 3 to 5 m. thick with a good roof and relatively horizontal, following somewhat the

contour of the surface. In the highly inclined thicker beds, filling methods were used; the workings in some places were bord and pillar with filling and in others alternate slicing and filling. The filling material at Trbovlje, Slovenia, the largest producing group of mines in the kingdom, was crushed limestone mixed with yellow clay. At the Brza Planka

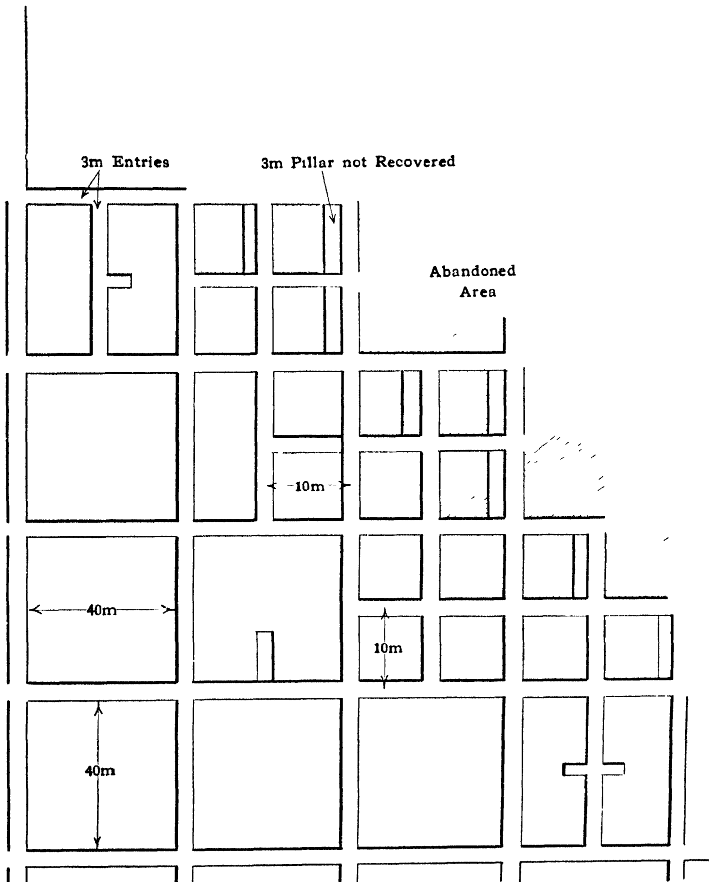

FIG. 7.—PLAN OF WORKING IN BOGOVINA MINE, SERBIA.

mine, in Serbia, a subsidiary of the Danube Steamship Company, clay was shoveled into the workings from which the coal had been extracted. The work of withdrawing pillars, in many mines, was so prolonged that great effort was necessary to resist the overburden pressure, instead of following the American practices of trying to keep in advance of the excessive weight.

There were no longwall mines in Jugoslavia. In a government-owned mine at Kreka, Bosnia, a slicing-and-caving method was used. The coal bed was about 20 m. thick and dipped from 18° to 20°. The coal was relatively free from impurities. It was mined from the top in slices about 7½ m. thick; on each 7½-m. level, 3-m. entries were driven across the bed at intervals of 30 m. and a series of 4-m. entries with 4-m. pillars were driven along the strike. Then entries were driven up the foot wall at intervals of 3 m. to the top of the bench or to the caved ground above the foot wall. The 7½ m. of coal was then mined retreating. The hanging wall consisted of about 20 m. clay, which caved with the mining of the coal. After a bench of coal had been removed, about a year elapsed before the succeeding bench reached the same place. During this time the clay in the hanging walls became consolidated and the coal removed in order. Very little timber was required for this mining.

George S. Rice (author's reply to discussion).—Mr. Scholz's plan of laying out and extracting pillars at the Valier mine is an advance in Illinois methods inasmuch as the pillars are wider than the rooms. On the basis of the dimensions of rooms and crosscuts given, exactly 50 per cent of that thickness excavated, that is, exclusive of roof coal, would be taken out by the first mining. Although Mr. Scholz states that the exact manner in which the pillars will be drawn had not been determined, he suggests slicing off at the face on the retreating plan. It is questionable whether it would be possible to slice off the ends of the pillars at the goave side efficiently, on account of the width of the rooms and crosscuts (25 ft.) and also as the pillar drawing is not to be brought back on a straight stepped line across the panels, if Fig. 6 correctly indicates the method. That is to say, with a system of pillar drawing in separate panels, with some strong rock in the strata overhead, the roof will tend to make a sudden slumping off, which might not occur if the retreating face could be brought back in a long diagonal line taking out panel pillars as well as room pillars.

It is true, as Mr. Harrington says,that little explosive is used or needed in blasting coal in most European mines, on account of the heavy overburden and numerous slips or cleats in the coal, often close together and generally inclined to the plane of the bed, so that the heavy weight tends to squeeze out the coal readily when it is picked or barred along the cleat. In many places, a straight pick mounted in a hammer air drill held against the face will bury itself in a few seconds in the coal, and the miner using this with a sweeping motion, as a man would use a scythe, readily slabs off the coal. There are, of course, exceptions in the coals closer to the surface, which are relatively hard, especially in the more regular beds of British fields.

It is true that a great deal of the coal is produced in finely divided form in Continental mines, leading to the extended use of briquetting, although considerable lump, termed "large" or "round," is produced in the British fields. The fact that the coal, as produced, is more broken than that from American mines calls for more elaborately planned screening and cleaning plants, and as the coal from any one shaft may come from several beds of different kinds, the coal from each bed must be handled and sorted separately. This increases the complexity of the surface plants and hence the number of men or women employed in coal dumping and screening.

Undermining machines have not been so freely introduced in Europe as in America, partly from inertia where they should be used in the harder coal, but principally because there is not the same need. However, loading conveyors of the fixed or shaking type are extensively used in the pitching beds; and in Great Britain chain conveyors are more extensively used, in the longwall or semi-longwall faces, than in the United States. One obstacle to the more systematic introduction of coal-cutting machines is that they must be air driven, although in Great Britain explosion-proof electric-motor driven machines may be used. The use of electricity, which is so universal in the collieries of the United States, is seriously limited by law in the European mines; except for the explosion-proof motors in Great Britain, it cannot be used except for fixed pumping installations and hoists, and slope engines especially installed on the intaking air currents. The trolley locomotive is barred, although it is one of the greatest factors for economy in American mines and it probably could be safely used on roadways with intaking air currents to great advantage in some of the European mines. American coal-cutting machines supplied with explosion-proof motors have been introduced in England but, except for air drills employed in tunnel driving, only hammer picks are used to any great extent.

The only mining machine that has been used, other than those of American origin or design, is the British disk longwall machine, which is employed chiefly in the comparatively level Yorkshire field. These are air-driven disks 6 or 7 ft. in diameter with cutters mounted on the rim. The British operators using these claim that they are more rugged than the American chain cutters and will stand better the work of cutting in the hard clay. This is questioned by American machinery representatives, but the writer has no information as to reliable comparative tests.

The mining conditions, in general, do not favor such extensive use of undercutting machines as do the flat coal beds of the United States, from which the bulk of American coal comes. The situation may be compared with that of the anthracite and bituminous districts of Pennsylvania, the Pittsburgh district being typical of the generally favorable American conditions and the anthracite district, with its steep pitching and faulted

coal, being comparable with the European conditions; although in Belgium and France the faulted conditions are much more severe. In the Pennsylvania anthracite district, where most of the operators readily adopt any promising means, some mining machines are used, but only a small portion of the output is machine mined, compared with the bituminous districts where the regular natural conditions favor the introduction of machines.

While the restrictions of mine labor are serious, as regards minimum wages and control of the industry in England, I do not think that the limitations at the face of the mine are as serious as prevail in this country. As variations in thickness of coal or pitch or the presence of partings are so much more common in Europe, it has been accepted as a matter of course that piecework prices must be adjusted from place to place in different parts of a mine, with certain minimum daily earnings guaranteed, or else the work in faulty coal is done by men on day wages.

Although the systems used in beds 5 to 10 ft. (1.5 to 3 m.) thick are much more expensive than the systems used in America, the average American mining method does not give a better total extraction; my observation is that quite the reverse is true. However, the bord-and-pillar systems, which are equivalent to longwall retreating, would be barred in many districts of the United States by the high yardage rates. It is regretable that in some districts in the United States, driving entries and rooms under so-called "deficient conditions" is more profitable to the miner than driving under normal conditions of the bed; and the yardage rates in many districts prohibit, from a competitive standpoint, the use of narrower rooms or entries where such would be better practice. The contrast is noticeable between the narrow room and thick pillar system of the Connellsville district, with its high recovery, and that of other mining districts in the central west where the recovery is less than 50 per cent. Mr. Scholz' plan is similar to the bord-and-pillar system in the laying out of rectangular blocks but differs materially in effect, as the rooms and crosscuts are twice as wide as those in the European bord-and-pillar systems.

No difficulty is experienced in keeping the sand filling separated from the coal later mined out by an adjoining slice. Moist sand stands very well; moreover there is usually enough clay in the sand to tend to bind the grains, so that with the support of some lagging no difficulty is experienced in getting out the coal up to the sand filling.

The thickest European coal beds in which the writer observed the mining methods were: at Weymss, Scotland, where a 24-ft. pitching bed, which contains partings, is mined by panel longwall successively in three advancing slices and one retreating slice; and at the Wykén colliery, near Coventry, where a bed about 30 ft. (9 m.) thick was mined in four slices by a retreating longwall method, retreating up the pitch. In

Upper Silesia, at Myslovitz, a 45 to 60-ft. bed was being mined by advancing longwall in successive slices parallel with the plane of the bed, with the aid of sand filling.

As to the question: Has Europe designed any method of mining whereby more than 75 per cent. of thick seams (say 12 ft. or over) can be extracted? I can say "yes." I have already referred to cases where the conditions are favorable, or where, as in Upper Silesia, an adaptation of the Pennsylvania system of sand filling is used.

As to the question, "What European practices can be adopted to advantage in the coal mines of the United States and what practices should be avoided?" First, to introduce the same amount of engineering skill and forethought in planning and developing coal mines that the Europeans have been forced to use from the difficulties of natural conditions, and which we use in the best mining practice in this country and especially in the large metal mines. The strongest criticism that I would offer on the majority of coal mining operations in this country is, that we apply the uniform methods of a district, fostered by an attempted uniformity of details in labor contract agreements, to specific mines; and if the mining conditions do not fit the system, as in a thin and faulty bed or where the coal has partings, it is either not worked at all or is worked in a most wasteful manner.

In general, in the European practice, the working conditions of each mine or part of a mine are studied so as to lay out and develop it to the best advantage. Heretofore, at least, the method of equitable payment for the miners has been sufficiently flexible to permit settlement between the underground management and the miners directly concerned, without having to resort to a formal negotiation between committees of miners and operators' representatives. The general impression that an observer of American mines receives is that they are all laid out on the same plan; whereas in European mines he is impressed with the variety of methods used to fit the varying conditions.

Second, European practice is better than ours in respect to the use of brick, iron, or concrete shaft linings, the arching and lining of permanent levels, and packing the workings with dry packing or with sand filling and washery refuse, where such material is available. Admittedly gob filling is an additional expense to the cost of mining, but only by such filling is increased recovery of coal obtained; the serious problem of subsidence, which is just beginning to confront operators in this country, is also considerably lessened thereby, and with sand filling it is entirely obviated.

Third, an increased adoption of coal-mining engineering in its broadest sense, like the high-grade mining engineering of northern Europe, applied both to underground operations and the development and manufacture of byproducts. The latter refers to utilization not only of

coking coal but of various coals and lignites. In this country, vast areas of lignites have not been utilized and it is frequently stated that they can not be used while bituminous coal is relatively cheap; but in Germany, in spite of the extensive Ruhr bituminous resources, low-grade brown lignites immediately adjacent are developed to a high degree, the product being used for electric-power distributing stations and the making of briquettes, which are used extensively with bituminous coal.

Fourth, increase of safety precautions. The general European practice is better than in the average mine of the United States. This is especially the case in mines producing firedamp; not that many of the gaseous mines in this country are not well managed, but in Europe there is not the same hesitation about acknowledging that a mine is gaseous and putting in the essential safeguards. In this country, until a mine is officially termed gaseous, and that usually means it is very gaseous, many chances are taken in the use of open lights, employment of electric trolleys in entries that have some gas in the air current, or even the use of trolley locomotives and non-explosion-proof machinery in working places which, at times, contain gas in threatening amounts.

The features of European practice to avoid are the over-elaborateness of mining plants, with respect to output, and the lavish use of labor, which was initiated when common labor was cheap. The over-elaborateness of equipment is particularly marked in the former Prussian mines in the Saar that were operated by the government.

It is evident that in the districts described by Mr. Smith, while they attempted to follow the general practice of the more advanced coal-mining regions of northern Europe, the organization and carrying out of the work has been inefficient. The original paper and the present discussion, have reference to mining in Great Britain, France, Belgium, and Germany and the northern part of the old Austrian Empire, which produces over 90 per cent. of the output of Europe.

Pillar Drawing in Thick Coal Seams

By G. B. Pryde,* Rock Springs, Wyo., and R. M. Magraw,† Hiawatha, Utah

(New York Meeting, February, 1921)

In laying out a new mine, provision should be made for the ultimate recovery of as much coal in any given bed as is consistent with safety and economic mining. Though each mining district, if not each individual mine, has problems that make the adoption of any hard and fast rule impossible, certain basic principles are adapted to practically all conditions. One of the most vital is the provision for barrier pillars of ample dimensions to protect all haulage and permanent airways during the life of the property.

Where conditions permit, room or butt entries should be driven on the strike or slightly to the rise, and rooms turned at right angles to these entries. The width of rooms and room pillars is governed by conditions prevailing in the mines under consideration. A safe rule is: In advancing, extract not more than 25 to 35 per cent. of the entire seam; in exceptionally thick seams, considerably less than 25 per cent. should be taken.

Rooms usually vary in length from 250 to 350 ft. (76 to 106 m.) with a barrier pillar of not less than 80 ft. between the last crosscut in rooms and the back entry of the level above. All entries, rooms, and entry-and-room crosscuts should be driven on sights and room crosscuts should be driven at right angles to the direction of the rooms. In mines under excessively heavy cover or where roof conditions are bad, rooms should be driven narrow and a barrier pillar left at every fifth to tenth room. This barrier should be of such dimensions that one or more rooms, with pillars of usual dimensions, may be turned through it when the barrier is no longer required.

Two of the advantages of such a barrier are the protection of the panel against peaking of roof load during pillar extraction and the easy closing of the entry in case of fire.

Where economic conditions permit, it is preferable to follow the retreating panel system of mining. In the case of coal that outcrops, and even in cases where no outcrop is exposed, it is preferable to commence pillar extracting on the high side of the mine. Where the retreating

* Assistant General Manager, Union Pacific Coal Co.
† General Superintendent, United States Fuel Co.

panel system is used, rooms should be turned in blocks of five to ten on the inside end of the uppermost panel, and as soon as this block of rooms is completed, the pillars should be attacked and another block of rooms started on the same entry outbye. Until a break line of considerable length is established, it is well to hold back the turning of additional rooms on an entry, unless roof conditions are good and the structure of the coal and roof is such that danger from caving, fires, and squeezes is minimized.

When starting to extract a block of pillars, one pillar should be brought down to a line intersecting the entry at an angle of 45°. The second may then be started, and so on until the break line extends from the back entry to the barrier pillar or head of the rooms above. In the mean time, rooms may be started on the inside end of the entry below, so that they will be completed at about the time that the chain pillar on the first entry is attacked. As soon as the chain pillar on the first entry has been drawn back for 50 to 100 ft., one or two rooms on the second entry may be driven through the barrier pillar and pillar extraction on the second entry is begun. Not more than two rooms should be driven through the barrier pillar at any one time, and the break line on the first entry should be at least 150 to 200 ft. in advance of the break line on the second entry.

Where roof conditions permit, the track can be carried square across the pillar end and moved down as the pillar retreats; but where the roof is bad, it is necessary to crosscut the pillar and leave a small stump of coal to protect workmen from the caves. Most of this coal can be recovered before the final room settlement occurs, but usually a small amount is lost each time that the roof breaks. Where this plan is followed, room No. 1 extracts pillar No. 2, and so on; then the miners are not forced to work under the loose end, which would be the case if the tracks were turned in the opposite direction.

In mines where the retreating-panel system is not possible, the same general rules as to width of rooms and pillars, spacing of crosscuts, and plan of procedure for pillar extraction should be followed. In no case, except under dire pressure, should the extraction of pillars be commenced on an entry containing working rooms on the inby side; nor, unless ample barrier pillars are provided or some peculiar natural condition exists, such as a fault or thinning of the seam, should pillar extraction be started in the middle of the mine.

Plan Used by United States Fuel Co.

In Carbon County, Utah, the coal is from 7 to 30 ft. (2 to 9 m.) thick; the cover rises rapidly from the outcrop to 1500 to 2500 ft. The coal is bituminous and a fine steam and domestic fuel. For years the formation

was classed as Laramie-Cretaceous, but recently it has been reclassified as Mesa Verde. In the Black Hawk and Mohrland mines the coal is massive in structure, seldom showing either vertical or horizontal cleavage. It is extremely hard and dense, and breaks with a distinct conchoidal fracture. The bed is clean throughout. Overburden is composed of alternating layers of sandstone and shale; the immediate roof is a massive sandstone about 125 ft. in thickness.

The average dip of the coal floor is about 3 per cent. and entries are driven on sights at an angle of 45° to the strike. Room entries are driven 12 ft. (3.6 m.) wide and the rooms are driven 22 ft. wide with a 50-ft. pillar. Room crosscuts are spaced 100 ft. (30 m.) and driven on sights. A 50 to 60-ft. chain pillar, with crosscuts 125 ft. apart, is maintained between the main and back entries. A barrier pillar from 100 to 150 ft. thick, according to cover, is provided between the tops of rooms and the back entry of the level above.

The coal is machine-cut to a depth of 6 ft. and is shot to a 6-ft. height in the first bench. Five to seven holes, charged with an average of two- and one-half sticks of 1½-in. ammonium nitrate permissible powder, are required to break down the face. The practice is to keep back 30 to 40 ft. of the second bench at all times, not only to insure a more stable supply of coal for the loader but that shorter jack pipes may be used with the mining machines. The second bench is shot 5 to 6 ft. in thickness, making the ultimate height of room 11 to 12 ft. The remainder of the top coal, ranging from 8 to 15 ft. or more, is brought back immediately in advance of the pillar line.

Before pillar extraction was begun a series of parallel lines conforming to the 45° break line were drawn at 50-ft. intervals. Wherever one of these lines intersected a pillar, a vertical line was painted, and the lines were numbered consecutively outwards; each line carried its number wherever encountered, hence the relationship of any retreating face to that of all others could be told by noting the nearest numbered line.

Maps of pillar workings are extended monthly and the mine foreman is furnished with a blueprint of the pillar section drawn on a larger scale than the regular pocket map. On this, the exact relationship of retreating faces to the uniform break line is noted.

At first, progress was slow for only one pillar could be started; but by working two shifts the first pillar was soon brought down far enough to permit starting the second, and so on until the break line extended from the back entry of the level above to the chain pillar of the entry upon which the pillars were working. It was not considered advisable to carry a continuous break line through the pillars on two entries; hence when extraction was commenced on the second level an offset of 100 ft. was maintained between the two break lines. Rooms from the lower level are driven through the barrier pillar one at a time, as the line of

retreat above permits. This insures a large block of solid coal for protection should a squeeze start in either section. Work is so planned that the top level will be about finished when work is begun in the third, so that no more than two or three entries will be worked at the same time.

Machines were used at first but it was soon deemed too dangerous to undermine the coal either by hand or by machine. Top coal is kept about 20 to 30 ft. in advance of the pillar faces. Props 6 in. or more at the small end are set under the lip of top coal in order to prevent falls of coal that may have been loosened by shots or by strain. No effort is made to set break rows, the timber being used simply for its warning effect when the roof is working for a break. Much of the timber is salvaged, but the height is so great that a slight subsidence breaks props and renders them valueless, except for ties or cap pieces.

Whenever the roof breaks, pillar faces are invariably lost. Sufficient warning is usually given to permit the loading of loose coal and the removal of track and other material to a safe place. Caves seldom extend beyond the end of the pillar. Places usually settle within 36 to 48 hr., so that work can be resumed. If pillar faces are lost, it is necessary to drive crosscuts, starting about 10 ft. from the cave, before continuing retreat. Small stumps remaining after completion of crosscuts are robbed of as much coal as possible; if it is unsafe to remove all the coal, stumps are shot in order to prevent hanging of the roof. In many instances practically the entire stump is recovered.

Up to date the extraction has been very satisfactory. In one area of over 10 acres, in coal 30 ft. thick in places, with an average thickness of about 24 ft. and with about 2000 ft. cover, the extraction was better than 90 per cent. In another area of about the same acreage, with thinner coal and less cover, the extraction was about 85 per cent., the loss being occasioned by the necessity of leaving up some top coal to support a friable roof. In a third area of about four acres, with coal 20 ft. thick and cover increasing from the outcrop to a height of 800 ft., the extraction has been nearly complete.

As some uneasiness was felt with regard to the possible action of the roof, considerable areas were extracted clean, but the roof showed no tendency to cave even when props were pulled or shot. After a break line 200 ft. long developed and the space worked out was 150 ft. wide, caves from 5 to 10 ft. in thickness occurred. No further caving took place until the open area was approximately 250 by 200 ft.

In the Mohrland mine the roof arched above the cave, and large openings remained between cave and dome. The roof being massive sandstone, the fragments of rock were extremely large and mostly angular. After this cave, the roof continued to break at intervals of about 50 ft., the caves going higher each time; at a height of about 90 ft. the inspection party was stopped, but as shown by an electric hand lamp the cave seemed

to be completely choked about 50 ft. higher. It is probable that caving extended much higher than this, but as very little weight has been shown on pillars it is thought that the void has been closed. In the Black Hawk mine, a second cave broke through to the surface in about 100 ft., the crevice being about 50 ft. wide by 200 ft. long, and the miners worked for several weeks by daylight.

The roof over the pillar section of the Hiawatha mine is composed of shale, with a small seam of coal (resulting from a split of the main seam) about 20 ft. above. Caved material is composed of finely broken shale and coal, and apparently the cave chokes completely, as no undue pressure has been noted on the pillars.

In this method of mining absolute adherence to plan is required, and while there is no certainty that trouble will not develop, it is believed that whatever success has been attained has been due to this strict adherence. A uniform break line insures even distribution of load over all pillars and prevents the peaking of roof load on one or more places, as is the case where the break line is jagged and irregular.

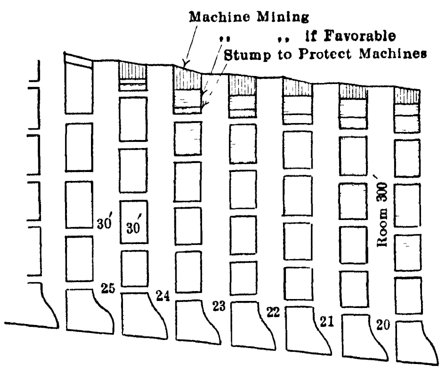

FIG. 1.

DRAWING PILLARS IN THE ROCK SPRINGS FIELD

In the Rock Springs field, the system is modified to suit conditions. The method used when the rooms are driven up the pitch is shown in Fig. 1. The rooms are driven for a distance of 300 ft. on 60-ft. centers; 30-ft. room and 30-ft. pillar.

When the boundary of the room is reached, a place 15 ft. wide is driven across the top end of the pillar on the strike of the seam, mining machines being used. When this block is taken out the remainder of the pillar is extracted by hand labor; the roof generally caves immediately after each successive block is removed.

Fig. 2 shows the method where rooms are driven across the pitch, the pillars being recovered in blocks of four. A breaking line is maintained by keeping the outside pillar in the lead. When roof conditions are good and cover is light, 75 per cent. of the pillar coal may be extracted with mining machines, the remainder by pick miners. In this case, a place 20 ft. wide is driven through the pillar, a 7-ft. stump

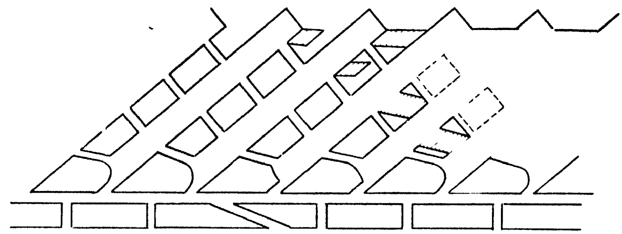

FIG. 2.

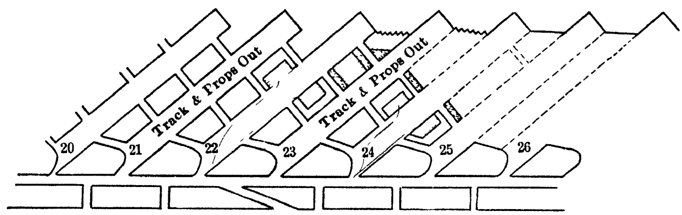

FIG. 3.

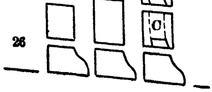

FIG. 4.

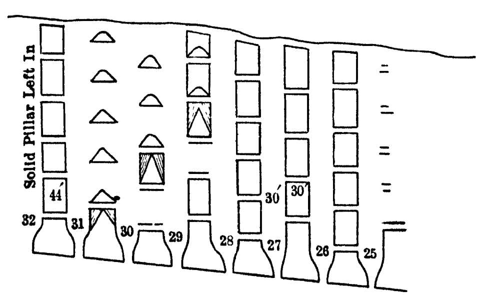

FIG. 5.

being left on the upper side. This stump is then recovered by miners as the machine is cutting another 20-ft. place below the first one, leaving a 7-ft. pillar as before.

In Fig. 3 the rooms are driven across the pitch but the props are drawn out of alternate rooms and a branch track is laid up the high side of the pillar. Both pillars are then recovered from the same room. This

system is only practicable when roof conditions are good, and lends itself readily to machine mining, like the system shown in Fig. 2. The system insures a good recovery of the coal, and all timber can be recovered in alternate rooms.

Fig. 4 shows a system that lends itself more readily to thick pillars. It is adapted to places where bad roof conditions exist. It also insures a rapid recovery of the pillar coal by working blocks A, B, and C, simultaneously. The mining machine cuts these blocks by running tracks in each crosscut; the small ribs are recovered by pick miners. This splitting system is generally followed in recovering entry pillars on the retreat.

In a method of drawing pillars in thick seams in which spontaneous combustion is apt to occur, no pillars are taken out until the boundary is reached; then the pillars are removed in blocks of three, a solid block being left at each sixth room so as to seal off in case of fire. Rooms are driven on 65-ft. centers when the cover is light, the rooms being 21 ft. and the pillars 44 ft. wide.

The work is started on the pillar nearest the face of the entry, three men generally working on each side of the pillar taking off successive skips, as shown, until the working parties meet in the center of the pillar; stumps are left to prevent rock slides when caving occurs. The withdrawal of pillars on the panel system is best accomplished according to Fig. 5, except that it has been found advisable to have the pillars on one panel retreat on a uniformly straight line. Much of the success of drawing pillars depends on systematic work along some pre-determined lines and the withdrawal of standing timber, when the roof is strong, is very necessary if successful results are to be obtained. To prevent crushing of the pillar the mining of the pillar coal should be started just as soon as the room has reached its boundary.

Advances in the Preparation of Anthracite

By Dever C. Ashmead,* Kingston, Pa.

(Wilkes-Barre Meeting, September, 1921)

Anthracite was first mined in the Wyoming Valley and sold as an article of commerce in 1808. As some preparation has always been necessary to make it ready to burn, the preparation of anthracite must date back over a hundred years. Two vital factors have determined to a large extent the degree and the method of anthracite preparation. These are, first, the character of the beds worked and the methods by which they are mined and, second, the equipment used and practices followed in the burning of the coal.

It is not the intention here to go deeply into these phases of preparation as a paper of no mean length could be prepared on the history of either. Rather, the intention here is to point out the main considerations and to show the influence they have exerted on the preparation of anthracite.

Mining Methods and Their Relation to Preparation

In the beginning of the nineteenth century, the coal beds were virgin with the possible exception of some outcroppings that had been worked, to a slight extent, by the Indians. It is known that the American aborigines had a knowledge of the use of this fuel, because when the Wyoming Valley was purchased from them, in 1754, they mentioned the fact that, by selling the land, they would lose their coal.

Real mining of anthracite began about 1808 when Judge Jesse Fell, of Wilkes-Barre, discovered that the "common stone coal of the valley" could be burned in an open grate. For some years thereafter mining was conducted near the surface. The working places were driven narrow; only the best of the coal was selected and the remainder was left in the ground. Only the lumps could be used, as no market existed for other sizes. As time passed, however, it became necessary either to go farther into the ground or to widen the working places. When these places were widened, falls of roof were apt to occur, making it necessary

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* Anthracite Editor, *Coal Age*.

to remove rock from the coal. This was particularly true of the Schuylkill region where the measures pitch steeply and the coal soon reached an excessive depth, judged by the standards of that generation.

At first only the thickest beds were mined, the thin measures being considered worthless; but as the thick seams were worked out, the thinner ones had to be opened. These carried a higher percentage of rock partings and bony coal, which had to be removed before the product could be marketable. This further complicated the methods of preparation. Later, particularly in the Wyoming field, a large part of the first mining in all the beds being completed, it became necessary, if the output was to be maintained, to commence second mining or the robbing of the pillars. The resulting falls of roof added more dirt to the coal, thus further complicating the process of preparation.

In the lower anthracite regions, where steep-pitch mining occurs and the coal is run out of the breasts into the mine cars by gravity, practically no way can be devised to keep the roof from falling and mixing with the mined coal. In many instances, the amount of coal and of rock in a mine car will be equal, while it is common for mined material to run steadily for some time at one-third coal and two-thirds rock. In the upper region, where the coal lies comparatively flat and the mine cars can enter the working chambers, the larger pieces of rock can be separated from the coal and a greater percentage of the latter loaded into the mine car. In the upper field, however, there is more second mining, or robbing, and the rock and bone that is now being produced from the thin beds to some extent compensates for the rock unavoidably brought down in the breasts of the lower regions. It can thus readily be perceived that because of the changes in the methods of mining and the exhaustion of the thicker beds the preparation of anthracite has become progressively more and more involved.

Methods of Consumption and Their Effect on Preparation

If coal were burned by the same methods today as it was 100 years ago, probably the changes in the methods of preparation would be less pronounced than they are. A tremendous advance in the methods of using the coal has, however, taken place. The types of furnaces have changed entirely, so that the coal must be prepared to meet the new conditions.

When Judge Jesse Fell conducted his experiments, he burned "the common stone coal of the valley" in a grate placed within an ordinary fireplace. It is natural to suppose that he used only the coal of larger size; 2 years later, at least, only lump coal was being employed for firing. Ignition may or may not have been secured by the aid of the small

hand bellows that formed one of the fire tools that reposed upon the hearths of that day.

Utilization of the finer sizes of anthracite was and still is largely a matter of support and draft. In the early days, grates were crude and of a wide opening. As the volume of the interstitial spaces is much greater in the larger grades, air will freely circulate through a thick bed of large-size fuel, such as grate, when it will move only with difficulty through a much thinner bed of, say, No. 3 buckwheat. When the first attempts were made to burn the buckwheats, it was found that natural draft would not cause sufficient air to move through the fire to maintain rapid combustion. Industrial plants, therefore, introduced forced or induced draft of some sort. The steam-jet blower was doubtless the first, the simplest, and probably the cheapest of these means of creating artificial draft; this was successful in a degree but was soon followed by the fan.

Combustion has been further perfected by placing the fuel on a traveling grate, and passing it successively over compartments carrying different draft pressures. These pressures are so arranged as to furnish the amount of air best suited to the fire conditions existing above them and to provide for complete combustion of the fuel before the refuse is discharged over the end of the grate into the ashpit. Already this arrangement is making present-day silt, or coal passing a $\frac{3}{64}$-in. (1.2 mm.) circular opening and containing probably at least 10 per cent. of moisture, a usable commercial fuel for power production.

Two other means of utilizing anthracite fines that are in limited use are briquetting and the burning of such fines in suspension. The first, while employed extensively in Europe, has never found great favor at the mines of this country. The product of the process is sold in direct competition with the domestic sizes of freshly mined coal. The second method has passed the experimental stage and is just entering that of commercial application. Time will determine the scope and possibilities of this means of anthracite consumption.

Preparation of Coal in the Early Days

From about 1808 to 1830, preparation—what little the coal received—took place in the mine itself. After mining, the miner loaded the larger lumps into a wheelbarrow and either dumped them directly into a canal boat or into a long chute by which they were conveyed to the boat. All sizes smaller than lump were left in the mine, being considered worthless. No power of any kind was used at the mines and all the work was to the rise, no pumps being employed.

In 1830, the first attempt was made to prepare the coal outside the mine. As before, only the large coal was removed to the surface, where it was dumped upon a perforated cast-iron plate. Here men with hammers

and picks broke the larger coal to commercial size. It is possible that after falling through the plate perforations the coal was passed over bar screens to remove the smaller sizes, which at that time had no commercial value.

About this time, also, the rake, Fig. 1, was introduced underground. After the coal was cut, the miner used a wrought-iron hand rake having teeth formed to give a clear opening of 1¾ in. (4.4 cm.) to separate all the coal smaller than stove from the larger pieces. The rake was used in parts of the coal fields as late as 1850, and perhaps later, for it was not until 1869 that it was found possible to use pea coal. Probably not until this size became a commercial coal was the rake discarded, but in some mines it was superseded by the slotted scoop shovel.

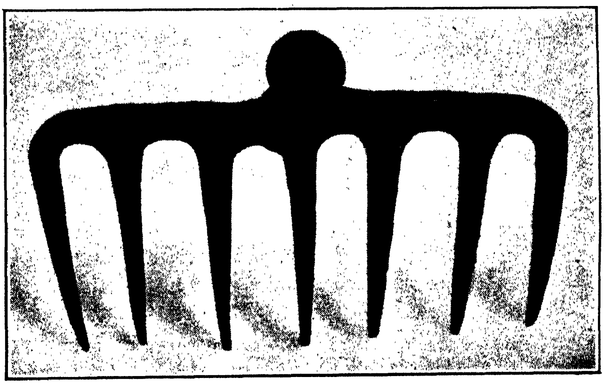

FIG. 1.—RAKE USED IN SEPARATING LARGE FROM FINE COAL IN MINES.

In 1844, J. & S. Battin, of Philadelphia, invented the roll crusher and installed it in their coal yard at that city. The same year the first coal breaker, with circular screens, was erected by Gideon Bast on Wolf Creek, near Minersville, Pa. A 12-hp. engine was used to drive the machinery and the equipment had a capacity for breaking and cleaning 200 tons of coal per day. The success of this building led to the construction of thirteen additional breakers in the same field the following year.

One of the earliest breakers, Fig. 2, of which a description exists occupied the same ground as the present Pine Brook breaker, in Scranton. Here, after the coal was broken down in the mine, it was raked over and the large pieces, or those over 1½ in. (3.8 cm.) in size, were loaded into the mine cars and hauled to the surface. The mine cars were run on to a trestle and discharged, by the aid of a horn dump, on to an incline. The coal was then pushed by hand or a rake on to a cast-iron plate, which

was so perforated that grate coal would pass through. On this plate the large lumps of coal were broken to smaller sizes, the lump and steam-

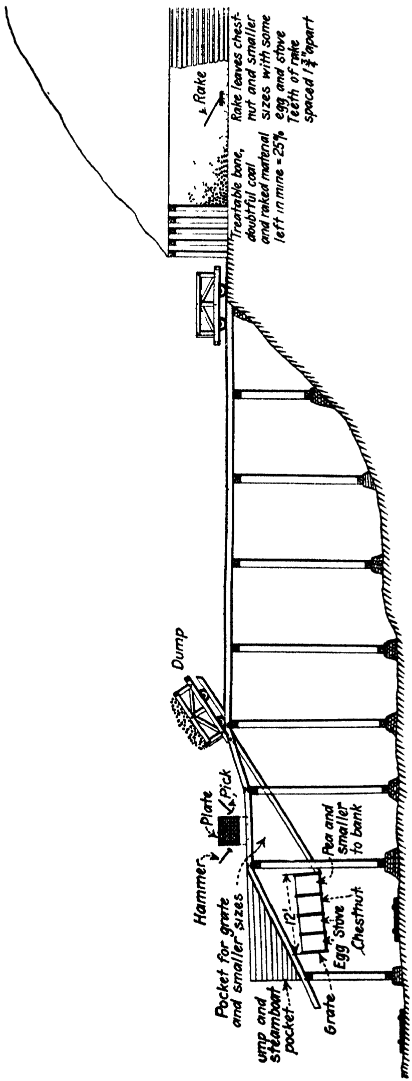

FIG. 2.—CLEANING COAL AT OLD PINE BROOK COLLIERY, SCRANTON, BASED ON TESTIMONY IN ROYALTY TRIAL. DRAWING PREPARED BY MR. DODGE, WILKES-BARRE, PA.

boat being pushed across the plate to a pocket. The smaller sizes fell through into a hopper, from which they were fed to a revolving screen 12 ft. (3.6 m.) long. The grate, or what is now known as broken coal,

passed out at the end of this screen and the rest of the material was separated into egg, stove, and chestnut. All coal finer than chestnut was run to a waste pocket and was transported from that point to a culm bank. This screen was revolved by man power, no mechanical energy of any kind being used in the preparation of the coal. The only impurities removed were those that the men took out when they broke the coal down through the cast-iron plate; the impurities were probably few. It is probable that this breaker was built about 1845, and was ten years old at the time it was described.

Perhaps the greatest improvement of this period was the introduction of mechanical power. This made it possible to use larger screens as well as to break down the lump coal by means of rolls, greatly increas-

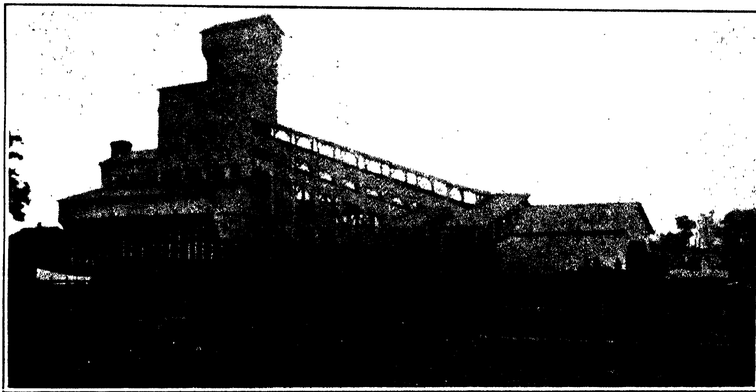

FIG. 3.—OLD DODSON BREAKER, PLYMOUTH, PA., BUILT IN 1869 AND BURNED IN 1897.

ing the capacity of these breakers over those employing the older hand methods. The hand-operated breakers were not, however, immediately superseded, for until about 1860 some were in existence.

The first rolls were made of cast iron with cast-integral teeth. About 1876, a roll having a cast-iron shell into which steel teeth were driven was introduced. In some places, a fluted roll was tried. In addition to these, sometime prior to 1865 a fluke roll was used. The teeth of the roll, in passing through slots in a cast-iron plate, crushed the coal in a manner exactly similar to that employed when breaking it with a hammer through cast-iron plates as has been already described. This method of crushing created as much waste but was more rapid than was the hand methods it displaced.

This brings us to the late sixties. At the old Dodson breaker, at Plymouth, which was built in 1869, Figs. 3, 4, and 5, the lump coal was separated from the smaller sizes and run directly to the lump pocket. The coal that was large and not very clean, together with the fine

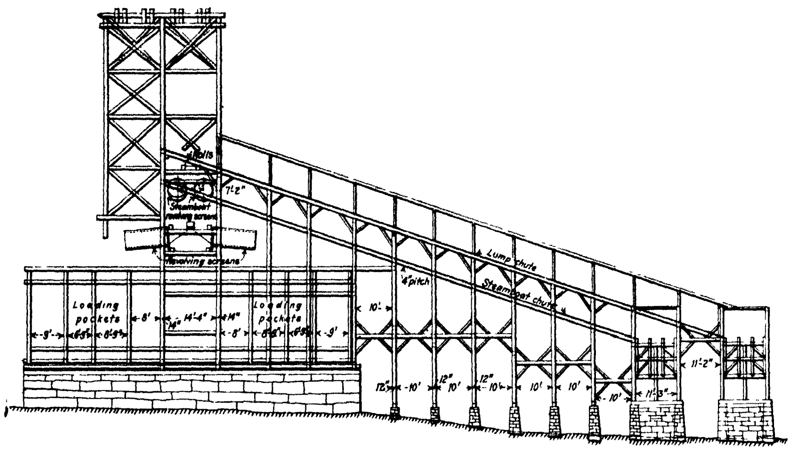

FIG. 4.—FRONT ELEVATION OF OLD DODSON BREAKER.

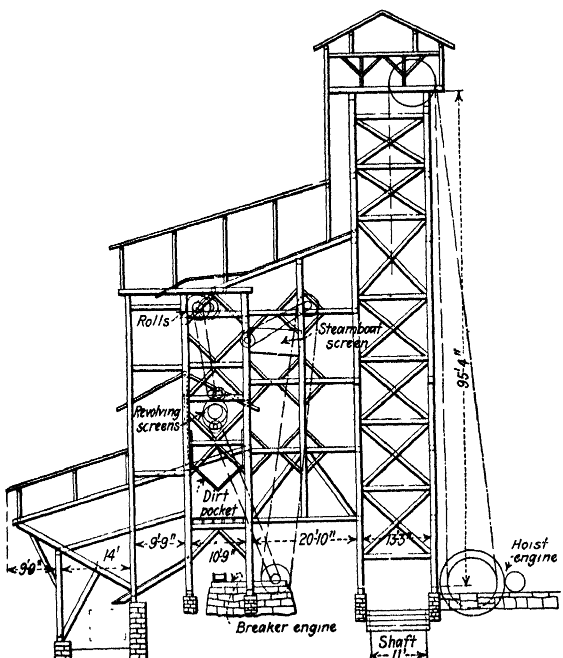

FIG. 5.—SIDE ELEVATION OF OLD DODSON BREAKER.

sizes, was passed to a pair of rolls and crushed; it was then separated into two equal parts, each part going to a separate revolving screen. In these screens only the steamboat coal was taken out, the finer sizes being sent to a second set of rotary screens. It is probable that the steamboat coal was hand-picked before being deposited in its pocket.

The finer sizes of coal that were separated in the second set of revolving screens were likewise hand-picked by boys before going to their respective pockets. No coal smaller than chestnut was saved, all smaller sizes, together with the rock, being sent to the dirt pocket and then removed to the culm bank. As the mine rake was still in use, all coal smaller than chestnut produced underground was left in the mine, so that the fine coal separated in the breaker was only what was made in the crushing down of the larger sizes; therefore, millions of tons of marketable coal now lie buried in the mines and will never be recovered.

Another interesting breaker of the same period is the old Washington or Reynolds breaker, at Plymouth, Fig. 6. This structure was reported as being in a dilapidated condition in 1869 but was braced and repaired so that it was used for many years afterward. It is interesting to note in connection with the accompanying illustration that 100 men were required to handle the output of this breaker. Of course most of the employees were boys hired to pick the slate and other impurities out of the coal. It did not seem to be the custom in those days to provide many windows to give light. Accordingly it was necessary to remove the roof over the picking chutes or leave them unprotected so that the boys could see to pick the slate. Fig. 7 shows the new Nottingham breaker to which the coal that formerly was treated in the Washington breaker now goes.

Another structure of the same period is the old Alexander Gray breaker, Fig. 8, which stood near the present Hollenbeck breaker in Wilkes-Barre. This was built in 1860 and was torn down in 1874. It was equipped with one set of revolving screens having a total length of 21 ft. 10 in. (6.7 m.) making culm-bank coal, small and large stove coal, egg, and No. 1 broken. Evidently the larger sizes were separated by hand.

Fig. 9 shows the old rolls that were discarded at some time prior to 1874. They were lying in a scrap heap when Mr. Dodge, a consulting engineer of Wilkes-Barre, made measurements of them and drew the original of which this illustration is a copy.

About 1870, the picking table was introduced, at the Hill & Harris colliery, Mahanoy City, Schuylkill County, after a series of experiments made by this firm.

The next important improvement in the preparation of coal was the introduction of the jig into the lower anthracite regions in either 1871 or 1872. The jig did not force its way into the Wyoming field

FIG. 6.—OLD REYNOLDS, OR WASHINGTON, BREAKER NEAR PLYMOUTH, PA.; THIS BREAKER WAS IN A DILAPIDATED CONDITION IN 1869.

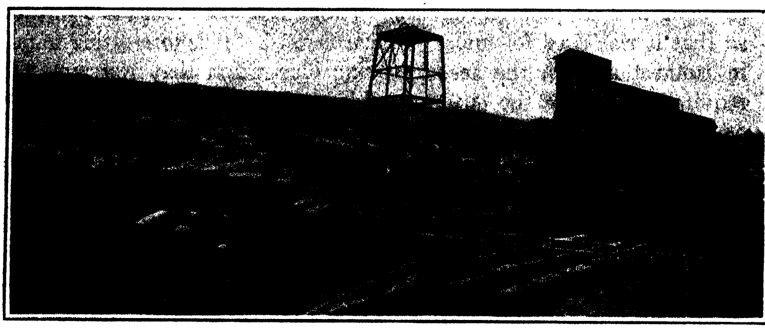

FIG. 7.—NEW NOTTINGHAM BREAKER OF THE LEHIGH & WILKES-BARRE COAL CO.

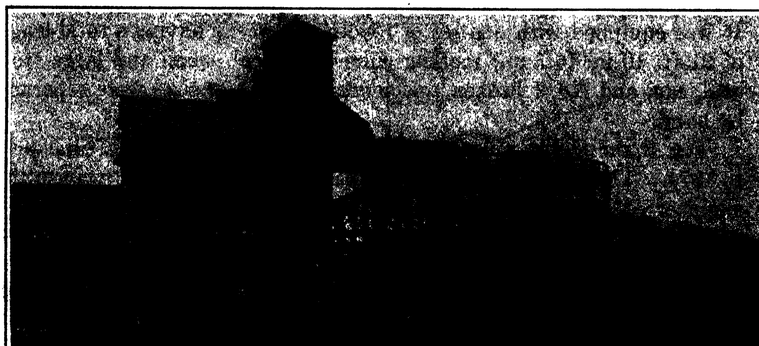

FIG. 8.—ALEXANDER GRAY BREAKER, WHICH STOOD IN WILKES-BARRE NEAR PRESENT SITE OF HOLLENBECK BREAKER. BUILT IN 1860 AND TORN DOWN IN 1874.

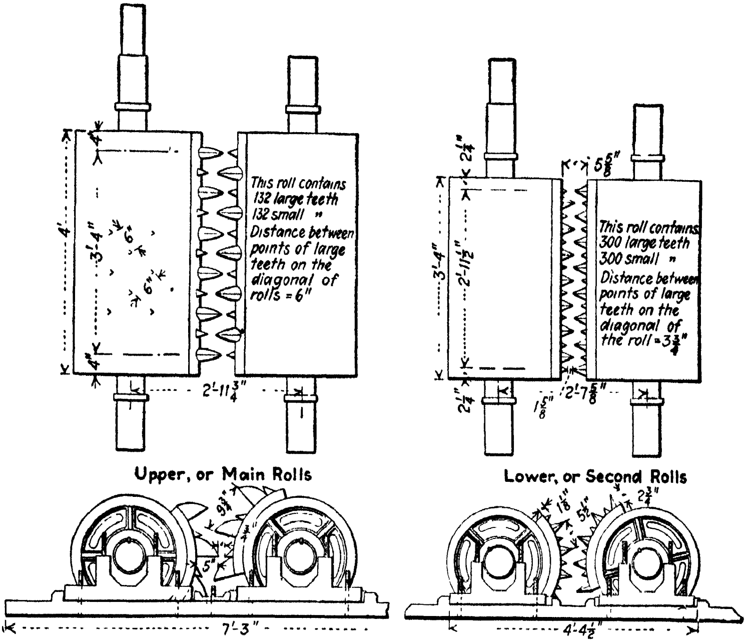

Fig. 9.—Old rolls used at breaker of No. 5 colliery of Lehigh & Wilkes-Barre Coal & Iron Co., prior to 1874.

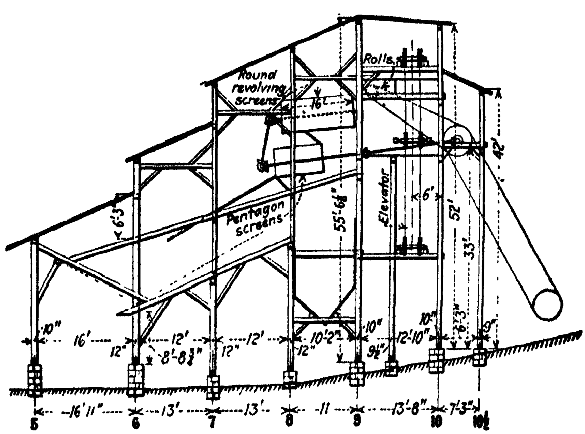

Fig. 10.—Section of Old Forge breaker of Pennsylvania Coal Co., built prior to 1884. Elevator was installed that year.

until a much later date because the coal in the upper region was much cleaner and drier than that produced in the middle and the lower fields, and the operators did not look with favor upon the wetting of their coal. In the lower region, conditions were entirely different. The coal came from the working places wet and so completely covered with fine mud that it was almost impossible to tell which was coal and which slate until the fine material and mud were washed off. It was therefore thought better, if possible, to separate the slate when the coal was being washed clean.

From 1872 until some time in the 80's no important inventions were made in the preparation of anthracite, but the existing types of machinery were improved and better results were obtained. In 1884, the Pennsylvania Coal Co. built the Old Forge breaker, which was one of the best equipped at that time; Fig. 10 shows the breaker as it was designed. In this breaker, the coal without separation was passed through the rolls, going thence to two sets of revolving screens, where it is presumed that lump, steamboat, and broken coals were separated, the fine material passing through to the pentagon screens below, which had the same diameter as the first set and were 10 ft. long. This breaker is the first in which the writer has found that the pentagon screen was used although it may have been used before this period. This type of screen did not prove satisfactory as it entailed too much breakage; however, there is one set of these screens still operating in a breaker in this region.

The Pennsylvania Coal Co.'s old No. 8 breaker at Dunmore, Fig. 11, built in 1889, embodies some new features. The coal was brought over a trestle, in mine cars, to the top of the structure and dumped directly on to a set of bar screens, which separated two sizes from the rest of the coal, presumably the lump and steamboat. The finer coals fell through the bars to a set of revolving screens below. Unfortunately, what sizes were prepared is indeterminate, but nevertheless some of the coal was taken in an elevator to the top and front of the breaker and there prepared.

The lump and the steamboat coals were passed over picking chutes where the rock was removed and run to the rock chute. The lump coal went either to the lump pocket or to the main rolls and was crushed, thence passed to revolving screens for further sizing. The coal from the screens was passed through chutes, where the slate was removed by boys. The steamboat coal seems to have gone to a set of pony rolls to be recrushed, as no pocket seems to have been provided for it. From the pony rolls, the coal went to a revolving screen and was sized. At this breaker the following sizes were made in 1889: Lump, broken, egg, stove, chestnut, pea, buckwheat, and bird's-eye. It is probable that the bird's-eye coal was the same as present-day rice, or No. 2 buckwheat. All coal finer than the bird's-eye went to the bank.

In No. 8 breaker, the effect that had been obtained by raking the coal in the mine was produced on the surface by the use of machinery. The fine sizes that needed no crushing were removed from the coal as soon as it was dumped, the large sizes alone being sent to the rolls for crushing. In this breaker, all the revolving screens were driven by gears, the rolls only being driven by belting. Here also the pentagon screen was employed. No provision was made to store coal at the head of the breaker so as to provide for a regular feed to the screen bars or other devices. The coal came at such irregular intervals as to interfere greatly with its preparation.

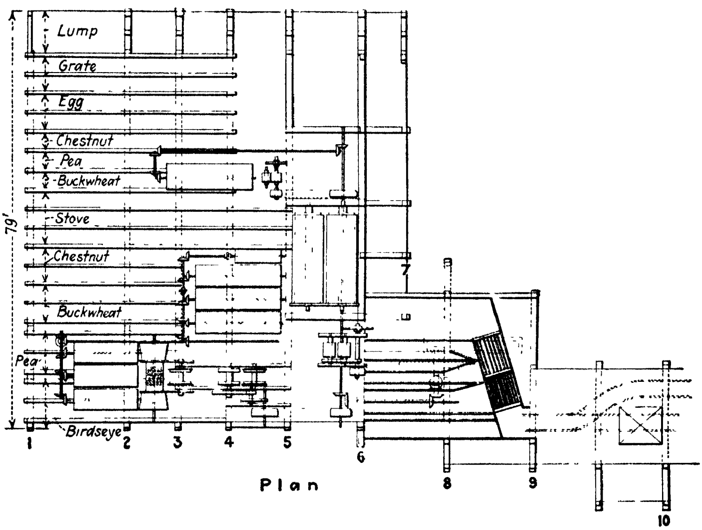

FIG. 11*a*.—OLD NO. 8 BREAKER OF PENNSYLVANIA COAL CO. AT DUNMORE, PA., IN WHICH THE COAL WAS SIZED BEFORE IT WAS SENT TO THE ROLLS.

Although this breaker was built when shaker screens were making their appearance they were not used, showing that they were not then considered sufficiently perfect to warrant their installation.

Just previous to the introduction of the shaker screen, Eckley B. Coxe, of Coxe Brothers, Inc., of Hazleton, invented the gyratory screen which this firm used for a number of years. These screens were satisfactory as to sizing and capacity, but their maintainance cost was high an account of the unbalanced vibration.

The Anthony shaker screen was among the first built, but this was preceded by a shaker that was supported on rollers and operated at an extremely high speed. The Anthony shaker was hung by wrought-iron rods. The suspension members were fastened to the shaker by a pin

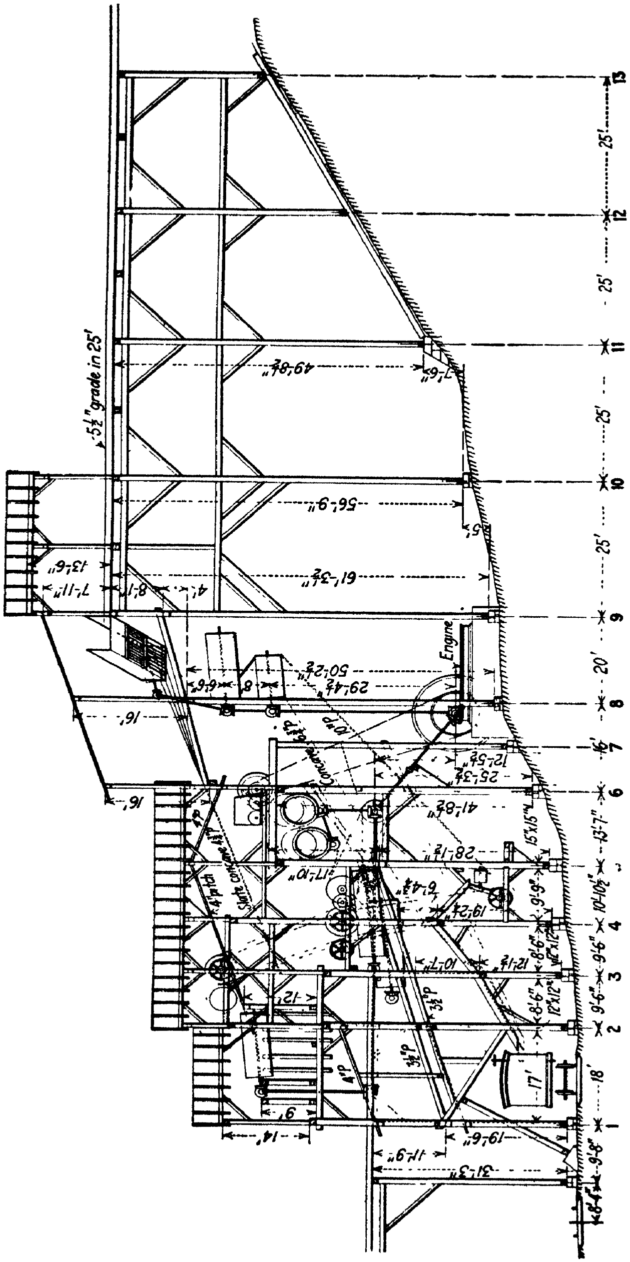

FIG. 11b.—OLD NO. 8 BREAKER OF PENNSYLVANIA COAL CO. AT DUNMORE, PA., IN WHICH THE COAL WAS SIZED BEFORE IT WAS SENT TO THE ROLLS.

and the top connection was a ball-and-socket joint instead of being attached rigidly, as is the Parrish shaker, which is the one in common use at the present day. The present type of shaker was not fully developed until 1907.

About the same time that the shaker was introduced the mechanical picker was invented, which revolutionized breaker design. Many types of these machines have been designed. The Zeigler picker, built about 1890, was the first of the "jump" pickers, and from it the various other designs involving the same principle have been evolved. This device

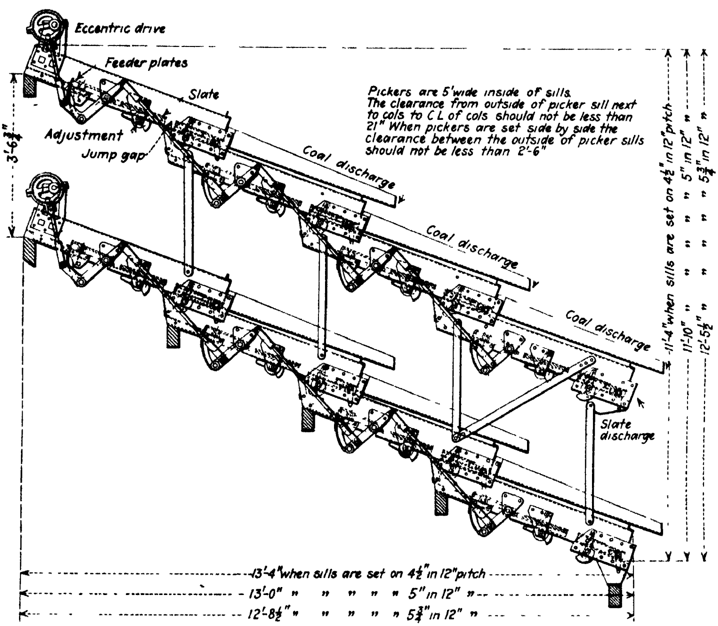

FIG. 12.—DEVERS PICKER, ONE OF THE SLOT-TYPE PICKERS, DIFFERS FROM THE OTHERS IN THE METHOD OF OPERATION. THE FEED IS RAISED AND LOWERED BY MEANS OF A CAM AND THE ANGLE OF THE SLATE IS LIKEWISE ADJUSTABLE

was followed, about 5 years later, by the Thomas picker, in which the mine product was passed over a slate slab. Friction between this slab and the slate in the coal was of course greater than between the slab and the coal. The slate accordingly was retarded as it passed through the picker and fell through the slot, whereas the coal leaped over it. The Thomas picker had only one slot. The next year, the Emery picker, which worked on the same principle but was a multiple-slot machine, was introduced. This device has been improved from time to time and now can be regulated to handle widely differing materials. Other inventors have worked out pickers operating on the same principle as the Emery

but embodying various improvements. Among these is the Devers picker shown in Fig. 12.

Many devices that will remove flat slate from coal have been invented. Among them is the Mowery, which is now in use at the Kingston Coal Co.'s No. 4 breaker. This picker operates as a shaker. The bottom plate is cut in a number of places, about half way across, and the edges of these cuts are turned down making slots. As the coal and slate pass over this plate the pieces of coal, being thick and rounded, do not pass through the slots but roll over them. The slate, however, when it reaches these slots tips up and slides through them. Of course any flat pieces of coal will naturally pass through the slots along with the slate. It is necessary therefore to pass the flat and the rounded product of the picker over some other type of picker. This machine was invented in 1905. Another flat-slate picker was invented by Colvin in 1910.

The Norman flat-slate picker consists of a number of rollers revolving in the same direction and so spaced as to permit the flat pieces of slate to pass through while denying passage to the rounded coal. The slate is thus caused to tilt and drop through the openings. The rollers are sufficiently inclined to cause the coal to move across them endwise. These pickers are still used to some extent, but as all mechanical pickers seem to be going out of use they also are passing away.

Another device of this kind that deserves mention is the Ayers picker. This consists essentially of a tilted traveling belt on to one end of which the unpicked material is fed, the belt traveling in an upward direction. The slate, being heavier than the coal and having a greater frictional resistance upon the belt, is carried up and discharged at the top, while the coal rolls down the belt and leaves it at the bottom. This picker is still used in a few places.

Probably the best known of all pickers is the anthracite spiral, which came into limited use about 1902 and into extensive use about 1904.

The Rolls

On the kind of rolls used and their operation depends, in large measure, the percentage of prepared sizes. Any excess production of the smaller sizes reduces the sum realizable from the output as a whole. Many types of rolls are now on the market and some coal companies have designed rolls for their own use. The main purpose of the designer is to obtain a roll that will crush or break coal to a certain size with the least amount of over- or under-size. All roll tests show the percentages of the prepared sizes of coal together with the smaller.

The accompanying set of curves, Fig. 13, was made from data secured in a long series of tests by the Lehigh & Wilkesbarre Coal Co. The rolls upon which these trials were made are of a design worked out by this company's mechanical engineering department and are particularly suited to its conditions. It will be noted that the amount of prepared

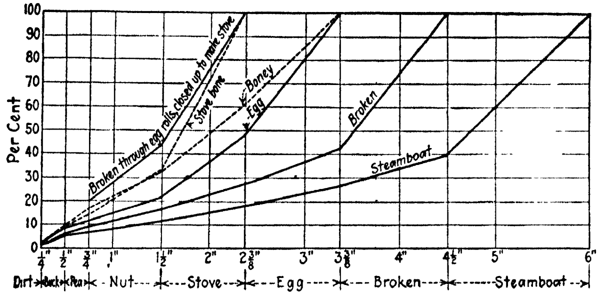

FIG. 13.—PERCENTAGES OF DIFFERENT SIZES OF COAL OBTAINED BY ROLLS AT COLLIERIES OF LEHIGH & WILKES-BARRE COAL CO.

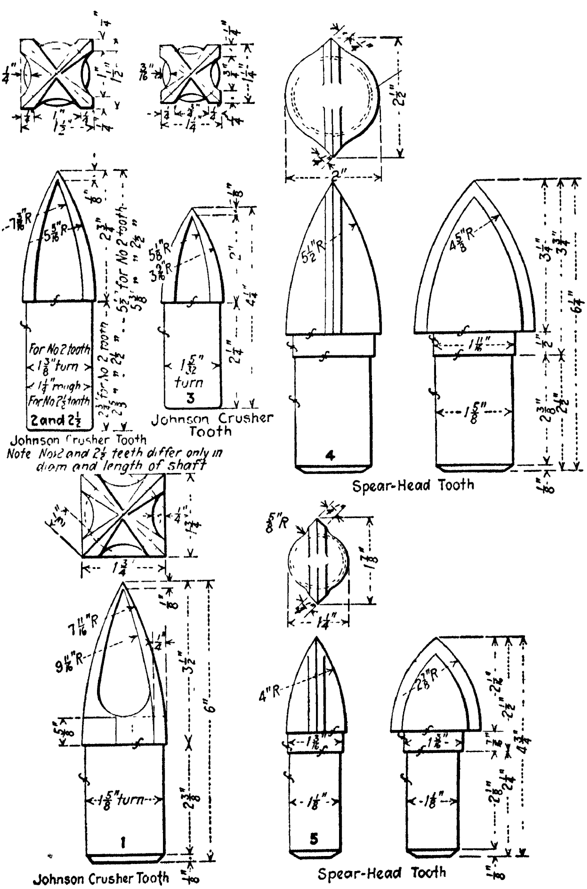

FIG. 14.—JOHNSON HOLLOW-GROUND TOOTH, COMPARED IN DESIGN WITH THE OLD-FASHIONED SPEAR TOOTH.

sizes of steamboat coal, that is, coal broken down from lump to steamboat, is 93.4 per cent., broken coal 91 per cent., egg coal 88.8 per cent.

Table 1 shows the results of a long series of roll tests made in different parts of the anthracite fields. The name or make of the roll, the locality where the test was made, the results obtained, and any special features of the roll are given.

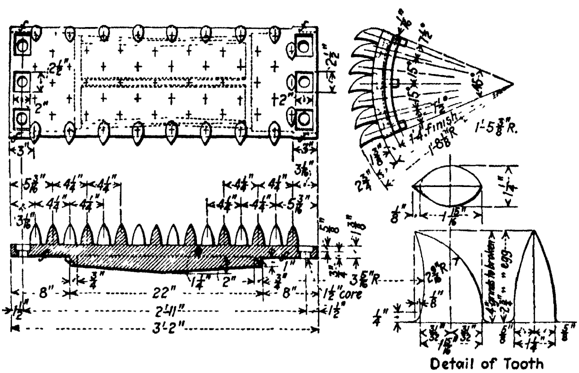

FIG. 15.—HAWK BILL TOOTH WHICH IS USED TO A LARGE EXTENT IN THE ANTHRACITE FIELD FOR THE CRUSHING OF COAL.

The Johnson hollow-ground tooth, differs from the old-style spear tooth in having four cutting edges spaced equidistant around its periphery, as shown in Fig. 14. Here is also shown the shape of the old type of spear tooth. In Fig. 15 is shown the "hawk bill" tooth which is of manganese steel. In Fig. 16 is shown one of the older type of teeth likewise made of manganese steel. The spaces between the cutting

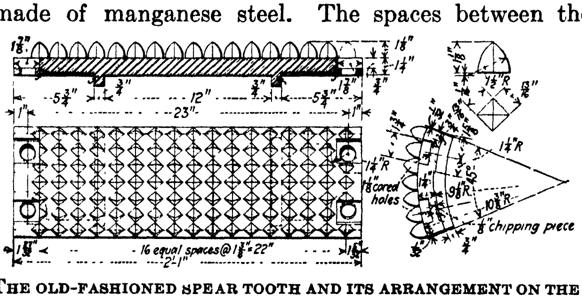

FIG. 16.—THE OLD-FASHIONED SPEAR TOOTH AND ITS ARRANGEMENT ON THE SEGMENTS.

edges of the Johnson roll tooth are concave, so that the only part of the tooth that comes into contact with the coal is the cutting edge; whereas with the older types, which possess only two cutting edges, the sides of the tooth present an unbroken surface to the coal, resulting in a grinding instead of a cutting action. It is also claimed that the wear on this type of tooth is such that the cutting edge is constantly maintained throughout its entire length of service.

Table 1 shows that in tests Nos. 26 and 27, made with Johnson teeth, the percentage of the prepared sizes secured exceeds the percentage made at the same colliery with another type of tooth. Furthermore the percentage of the larger sizes of coal is greater for the Johnson than for the other types of teeth. Test 35 shows that, while almost no broken coal was made by the Johnson tooth, yet the percentage of prepared sizes was greater than in test 34 when 36.4 per cent. of broken coal was produced.

A recent invention by Frank Pardee, shown in Fig. 17, is in daily service in the Coleraine breaker. This machine consists of two shafts, which are revolved toward each other by gearing. On one shaft is mounted a steel plate carrying cutter teeth on its periphery; these are similar to the teeth on a circular saw, but the distance between the adjacent cutters is equal to about the diameter of the pieces to which it is desired to break the coal. Thus an egg crusher would have teeth 2⅜ in. (6 cm.) apart. The other shaft carries plain steel disks. The cutters and the disks revolve in parallel and alternate planes, their tops moving toward each other.

The movements of the feeder and the cutter are synchronized, so that each piece of coal is delivered in front of a tooth, which carries the coal forwards against the opposite disk. This acts as a fulcrum, against which the cutter breaks the coal. Between adjacent cutters, as well as between the disk wheels, spring spaces are placed. Any coal that lodges between either the cutters or the disks will push back the spring, thus preventing the coal from being crushed. The coal is so fed to this device, by a new and ingenious spiral feeder, that each piece of coal is delivered with its major axis at right angles to the plane in which the cutters revolve.

The following is the result of a test run on one of these rolls:

	Per Cent.
Over 3¼ in	0.00
Over 2⁵⁄₁₆ in	71.42
Over ¾ in	21.42
Over ½ in	2 38
Over ¼ in	2.85
Over ³⁄₁₆ in	0 48
Over ¹⁄₁₆ in	1 07
Smaller	0.48

The material was crushed directly from broken into egg coal without any oversize, giving a total of 92.84 per cent. of prepared sizes. With the common roll, it is generally expected that the highest yield of prepared coal will occur when there is 15 to 20 per cent. of oversize made in the roll.

TABLE 1.—*Results of Tests at Various Breakers*

Name of Colliery	Name or Make of Rolls	Size	Special Characteristics	Size of Coal Crushed	Speed, Ft. Per Min	R.p.m.	Date	Steamboat, Per Cent.	Broken, Per Cent.
Short Mountain.	Solid cast	No. 2	Teeth diagonal and alternate	Broken	918	115	1910		
Williamstown...	Manganese	No 2	Alternate Teeth	Broken		80	1911		1.3
Williamstown...	Manganese	No. 2	Alternate Teeth Removed Each sheet	Broken		80	1911		9.0
Williamstown...	Manganese	No. 2		Broken		80	1911		7.0
Williamstown...	Manganese	No. 2		Broken		80	1911		3.8
Wm. Penn. ...	Manganese	No. 2		Steamboat	301		1920		43 0
Cameron	Manganese	No. 2		Steamboat	256		1920	12.5	38 8
Luke Fidler ...	Manganese	No. 2		Steamboat	233		1920	2.9	26.2
Slott......	Manganese	No. 2		Steamboat	364		1920		42.7
Pennsylvania....	Manganese	No. 2		Steamboat	345		1920		29.0
Wm. Penn	Manganese	No. 1		Lump	905		1920	51.0	21.0
Cameron	Manganese	No 1		Lump	250		1920	64.5	6.5
Luke Fidler. ..	Manganese	No. 1		Lump	1107		1920	38.1	16.7
Scott	Manganese	No. 1		Lump	1008		1920	30.0	21.9
Pennsylvania.	Manganese	No. 1		Lump	240		1920	36.0	20.5
Richards..	Manganese	No. 1		Lump	288		1920	43.3	17.8
Cameron.. .	Manganese	No. 3		Broken	246		1920		15.3
Luke Fidler. ...	Manganese	No. 3		Broken	281		1920		5.0
Richards	Manganese	No. 3		Broken	289		1920		35.0
Jeddo No 4 ...	Lloyd	No. 2		Steamboat and broken		135			
Jeddo No. 4......	Lloyd	No. 1		Lump		135		38 0	20.0
Highland No 5..	Lloyd	No. 1		Lump		135		41.8	29.0
Highland No. 5.	Lloyd	No. 2		Steamboat and broken		135			
Lansford No. 5 .	Lloyd	No 1		Lump		135	1920	41.4	19 8
Lansford No. 5	Lloyd	No. 1		Lump		135	1920	30.5	24.0
Lansford No. 5..	Lloyd	No. 1		Lump		135	1920	45.1	23 6
Lansford No. 5.	Lloyd		Johnson hollow ground tooth	Lump		135	1920	46.1	22.5
Lansford No. 6..	Lloyd	No. 2		Steamboat		135	1920	9.6	38.7
Lansford No. 6	Lloyd	No. 2		Steamboat		135	1920	0.6	43 6
Rahn No. 11 .	Manganese	No. 2		Steamboat		135	1920	21.0	35.4
Rahn No. 11....	Manganese	No. 2	Johnson	Broken		135	1920		26.0
Rahn No. 11....	Manganese	No. 2		Steamboat		135	1920		4.4
Rahn No. 11....	Manganese	No 2	Johnson	Steamboat		135	1920		5.9
Tamaqua No. 14.	Lloyd	No 2	Johnson	Steamboat		135	1920		36.4
Tamaqua No. 14.	Lloyd	No. 2	Johnson	Steamboat		135	1920		0.5

TABLE 1.—*Results of Tests at Various Breakers*—(*Continued*)

Egg, Per Cent.	Stove, Per Cent.	Chestnut, Per Cent.	Prepared Sizes, Per Cent.	Pea, Per Cent.	Buckwheat, No. 1, Per Cent.	Rice, Per Cent.	Barley, Per Cent.	Buckwheat No. 4, Per Cent	Total Small Sizes, Per Cent.	Number of Teeth Per Shell	Condition Teeth	Test	Remarks
47.5	21.0	13.5	82.0	6.0	5.5	3.5	1.75	1 25	18.0	896	Fair	1	
37.3	27.7	16.2	82.5	5.0	6.0	3.6	1.2	1.7	17.5	840	Good	2	
39.5	22.9	13.2	84.6	4.5	4.6	3 0	1.5	1.8	15.4	840	Good	3	
39.4	22.8	14.8	84.0	5.0	4.5	3.3	1.7	1.5	16 0	840	Good	4	
40 0	23.0	14 9	81 7	5.9	5.6	3.7	1.4	1.7	18.3	840	Good	5	
21 0	12.5	9.0	85.5	6.0	3.5	4.0	1.0		14.5			6	
12 5	13.7	9 2	86.7	4.2	4 3	2.0	2.5	1.3	13.3			7	
18.0	18.4	16 0	81 5	6.8	4.9	3.4	2.4	1.0	18.5			8	
19.0	12 7	11.7	86.1	4.7	4.2	2.6	1.5	0.9	13.9			9	
35 0	21.0	11.2	96 2	1.5	1.2	0.5	0.3	0.3	3.8			10	
8 5	6.0	4.8	90 3	3.0	3.2	3.0	0.5		9.7			11	
4.5	7.8	6.0	89 3	3 5	3.7	1.0	1.5	1.0	10.7			12	
12.4	9.5	10.0	86 7	6 0	3.3	1.9	1.4	0.7	13.3			13	
14.7	11.7	10 0	88 3	4.2	3.7	2.1	1.1	0.6	11.7			14	
15 3	11.3	10.0	93 0	3.0	2.5	0.8	0.5	0.2	7.0			15	
12 3	8 1	6.3	87.8	4.3	2.7	2.3	1.8	1.1	12.2			16	
38 7	18.0	15.8	87.8	4.5	2.2	3.0	1.5	1.0	12.2			17	
26 0	33.0	23.0	87.0	5.0	2.5	2.5	1 5	1.5	13.0			18	
27.7	15 6	11.1	89.4	3.8	2 8	1.9	1.4	0.7	10.6			19	
52.0	28.0	10.0	90.0	3.0	3.0	2.0	1.0	1.0	10.0			20	10 tests average
19.0	8.0	6.0	91.0	3.0	2.0	2.0	1.0	1.0	9.0			21	7 tests average
20.0	4.5	2.0	97.3	1.0	0.7	0.7		0.3	2.7			22	
63.4	19.2	9.6	92.2	2 8	2.0	1.7	1.0	0.3	7.8			23	
11.1	8.2	8.3	88.8	3.6		7.6			11.8			24	
15.5	9.2	10.1	89.3	3 8		6.9			10.7			25	
7.6	7.9	8.6	92.8	2.6		4.6			7.2			26	
9.6	4.9	7.3	90.5	3.4		6.1			9.5			27	
20.9	10.6	9.3	89.1	3.8		7.1			10.9			28	
27 4	9 8	8.7	90.1	3.4		6.5			9.9			29	
16 6	8.6	7.2	88.8	4 0		7.2			11 2			30	
39.0	15.3	10.6	90 9	3.9		5.2			9.1			31	
37.2	28 3	16.2	86.1	3.5		10.4			13.9			32	
51.8	20.5	11.0	89.2	3.8		7.0			10 8			33	
32.8	11.5	8.9	89.6	3.5		6 9			10 4			34	
54.8	22.8	12.0	90.1	4.1		5 8			9.9			35	

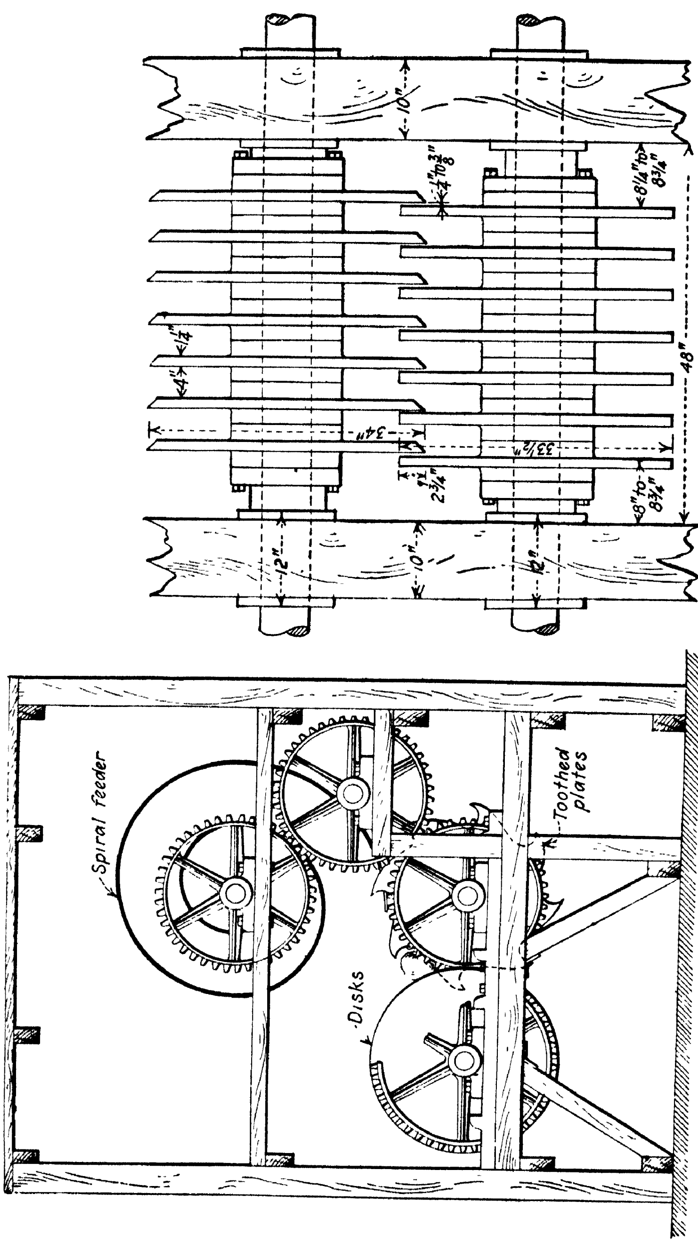

FIG. 17.—PARDEE ROLLS, AN ENTIRE DEPARTURE FROM PRESENT ROLL PRACTICE. THE COAL IS FED TO TEETH BY A SPIRAL FEEDER SYNCHRONIZED WITH TEETH OF ROLLS

Shaking Screens

The design of shaking screens has become so nearly standardized that few improvements have been made during the past several years.[1] But a few details will be given concerning the current practice at certain plants.

In the new Wanamie breaker of the Lehigh & Wilkes-Barre Coal Co., the following areas, in square feet of screening surface, are allowed for one ton of coal per hour; the large screen area is provided so that it is possible to handle and screen the coal properly during the morning when the tonnage for a short period is unusually heavy.

	Square Feet		Square Feet
Broken	2.4	Pea	6.4
Egg	1.9	No. 1 Buckwheat	4.2
Stove	2 3	Rice	6.9
Chestnut	2.3	Barley	7.8

The Susquehanna Collieries Co. uses the following areas:

	Square Feet		Square Feet
Egg	0 75	No. 1 Buckwheat	1.50
Stove	0 75	Rice	1.75
Chestnut	0.875	Barley	2.00
Pea	1.25		

In both cases the broken and egg coals are screened dry whereas the other sizes are screened wet.

Feeders

Under ordinary conditions, when the breaker begins operations in the morning there is an abundance of coal, produced possibly by the night shift or left over from the preceding day. This means that during the first few working hours too much coal may be presented for the breaker to handle properly; later, the rate at which the coal arrives decreases and the breaker may not receive its capacity supply. At times during the day, the delivery of coal may be irregular so that at one moment the preparation equipment will be overtaxed whereas a few moments later it may not have sufficient coal to operate properly.

These abnormal conditions may be met in two ways: First, by making the capacity of the breaker so great that the breaker will be capable of handling all the coal delivered to it. This method requires the construction and equipment of a building far beyond the average needs. The second method is to provide storage hoppers to bridge over the peaks. This is the better method, for then the breaker can be so designed that the cost of construction and operation may be reduced to a minimum.

[1] Paul Sterling: Preparation of Anthracite. *Trans.* (1911) **42**, 277.

In most cases these hoppers are placed at the top of the breaker; they may be so located, however, that both the coal passing through the bull shakers and that leaving the main rolls will go to them. In addition to this storage, many companies have arranged storage pockets in front of, and feeding, the jigs. If the coal comes too rapidly for the jigs to handle, it is placed in these storage pockets so that it can be fed as desired.

In order to accommodate a temporary excess of coal and assure its being fed properly from the hoppers or storage pockets to the breaker equipment, feeders are used. In some places these are operated by hand, but the more common method is to employ an automatic feeder. The number of openings and the quantity of coal admitted can be regulated to suit the conditions existing at the plant.

Feeders are of many types. The ordinary gate feeder, in which a gate is raised and lowered at suitable intervals, is objectionable as coal is liable to get under the gate, in which case it is crushed or it interferes with operation. But where the gate rises through the coal to shut off the flow and sinks below the pocket or chute floor to allow the material to flow, the gate does not break the coal. Feeders having a door opening outward and then closing have the disadvantage that a large lump of coal is liable to prevent the door from closing properly. Many varieties of the reciprocating feeder have been invented but they all operate on similar principles. Their advantages are many. The device is driven by an eccentric, which pushes the plate backwards and forwards; it usually operates at the bottom of the pocket. The coal feeds down to the feeder, which pushes it forwards on to the shaking screens or other piece of preparation equipment and then returns for another load. The revolving feeder is similar in appearance to a shrouded gear or pinion bearing, say four teeth and a corresponding number of feeding compartments. Coal runs into these compartments and is discharged as the feeder revolves. The rate of feed depends entirely on the size of the feeder and the number of revolutions made per minute.

Chutes

The three pieces of equipment, employed in the preparation of coal, where most degradation occurs are the rolls, the chutes, and the pockets. The general manager of one of the larger companies has said: "It is difficult to give figures on chute tests because of the varying conditions, but it is probably safe to say that the total breakage in handling coal from shakers to the lip screen at the loading pocket would be from 10 to 12 per cent." The loss from this cause during a year may be very great, for the prepared sizes of coal are worth, at the mine, from \$7.50 to \$8 per ton, while the steam sizes are worth only from \$1.50 to \$3.50 per ton.

The operators are using various types of chutes and chute linings in

order to decrease the breakage, reduce the space taken by the chutes, and increase the life of the chute linings.

At first, when the coal was prepared near the top of the breaker, it was carried to the pocket by long inclined chutes; in one breaker

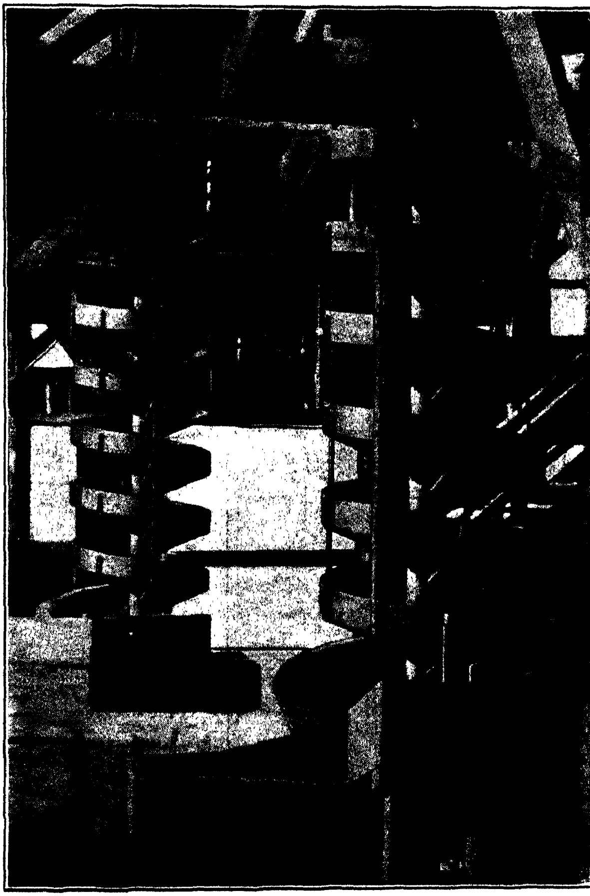

FIG. 18.—STEEL SPIRAL CHUTES USED TO LOWER COAL HIGH IN BREAKER.

one of these chutes was 350 ft. (106.7 m.) long. The friction of the coal on the chute lining and against the sides, together with poorly designed curves, caused much unnecessary coal breakage. To-day shorter chutes of different design are provided. Among these are the spiral cast-iron chute, the spiral steel chute shown in Fig. 18, and the spiral built-up chute shown in Fig. 19. The advantage of this type of chute is that it

takes up little space and the coal passes around curves so designed that the breakage is reduced to a minimum; the objection to it is that the acid water soon corrodes the lining.

A type that is coming into favor is the box chute, shown in Fig. 20. This is a square, verticle wooden box lined with sheet iron. The coal passes down an inclined chute leading to its top, in which is placed a pan that

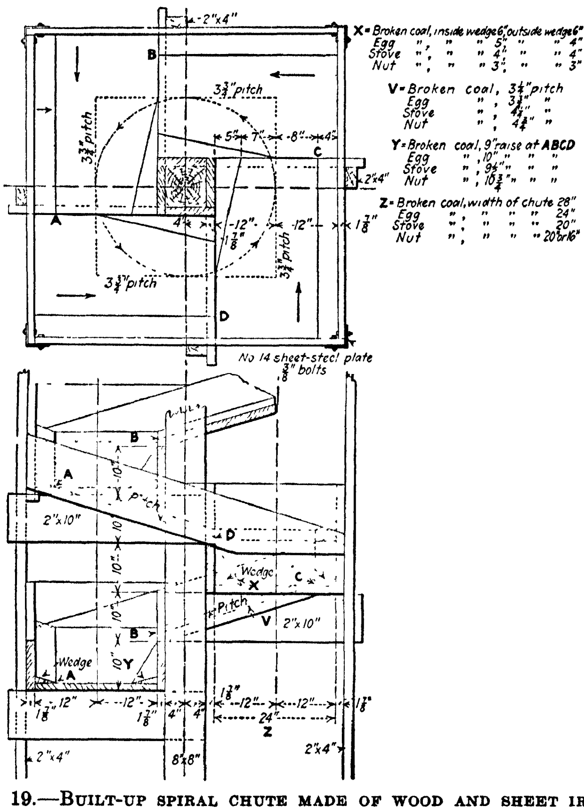

FIG. 19.—BUILT-UP SPIRAL CHUTE MADE OF WOOD AND SHEET IRON.

rises and falls in accordance with the amount of coal upon it. This pan is connected by a lever and wire rope to a door in the bottom of the vertical chute. When the pan ascends, the door closes; and when the pan is forced down by the weight of coal the door opens. The vertical box is constantly filled but as the coal backs up the inclined chute, the pan is depressed and the bottom door is opened, allowing the coal to pass out. When the weight of coal on the pan is not sufficient to keep it down, it rises and the door closes. In this way the box is kept full of coal.

As there is some breakage in the square corners, in the chutes now being built the corners are rounded. One disadvantage of the box chute is the difficulty of relining it; it is probable, however, that round vitrified clay pipe can be substituted for the box.

In many places a shaking chute conveys the coal from one point to another. The advantage of this construction is that the shaker can lie at a small pitch and reduce the vertical distance to a minimum. Retarding conveyors are also being used for lowering coal. By their use breakage is reduced greatly but they require power for their operation

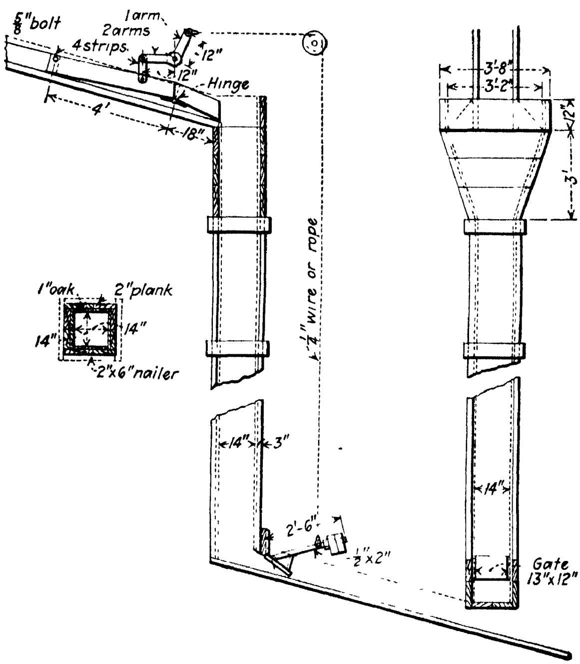

FIG. 20.—BOX CHUTE USED BY SUSQUEHANNA COLLIERIES CO. TO LOWER COAL IN BREAKER AND REDUCE BREAKAGE.

In the White chute, shown in Fig. 21, the coal is fed through a spiral chute provided with a control regulating the amount of coal that can pass. This arrangement is similar to the box chute but works in an opposite manner. The coal passes under the pan and operates the gate in the chute; it is then fed on to a movable chute, the end of which is close to the sloping bottom of the pocket. When discharged from the end, the material runs down the bottom of the pocket. As the pocket fills, the coal pushes back the traveling chute to offset the friction in the travel-

ing chute. Tests have shown that this type of chute reduced breakage in the pockets 5 per cent.

Renewal of chute linings is an important item in the cost of breaker operation. For years nothing but blue annealed sheet iron was

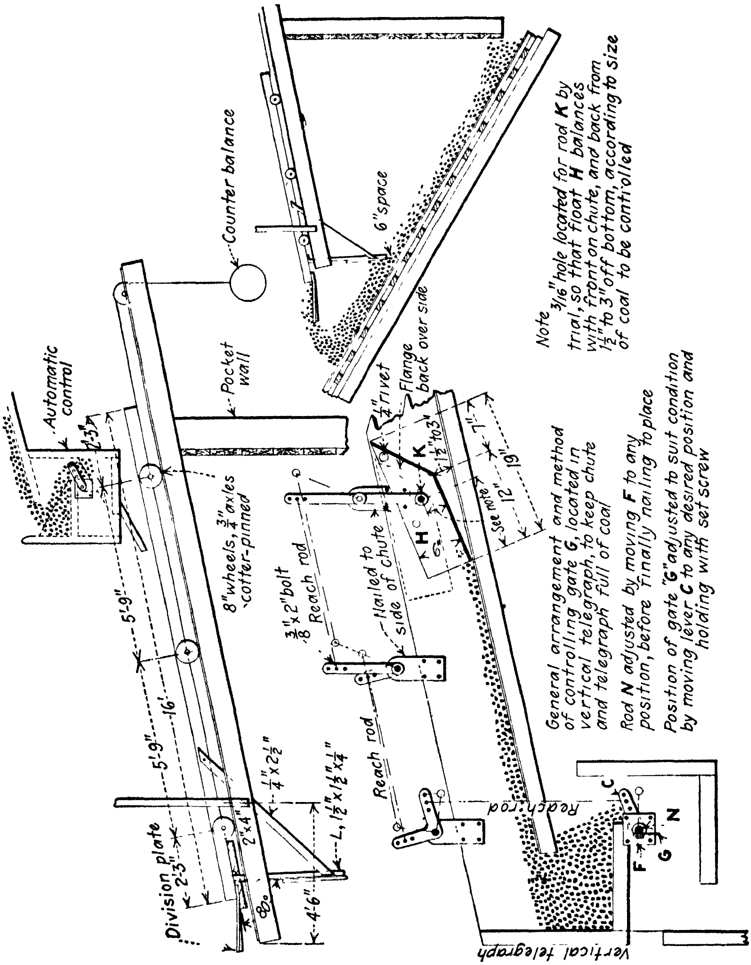

FIG. 21.—WHITE CHUTE FOR SENDING COAL TO POCKETS AND CONTROL TO REDUCE BREAKAGE. THIS CONTROL CAN BE USED ON ALL CHUTES IN BREAKER.

used; as the acid water soon corroded this a sheet seldom lasted more than three or four months. A number of companies using the old type of chute are lining them with galvanized sheet iron; this resists the action of the acid in the water and also permits the coal to slide on a smaller angle.

Vitrified-clay pipe makes an excellent lining for coal chutes and appears to resist wear indefinitely. Some chutes so fitted that have been in active service for more than 3 years show only slight signs of wear. Great care must be taken in the lining installation to see that the ends of the pieces of pipe exactly match, as otherwise they will wear out more rapidly than when set properly. Glass has been tried by some companies but is too brittle. Monel metal also has been used in a few places and to a slight extent.

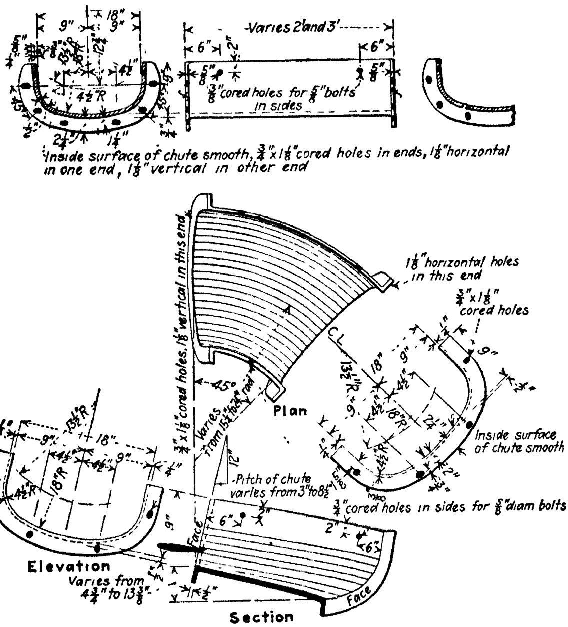

FIG. 22.—SECTIONS AND DESIGN OF CORROS METAL CHUTES NOW USED BY THE HUDSON COAL CO. IN THE MARVINE BREAKER, FOR BOTH STRAIGHT AND SPIRAL SECTIONS.

The Hudson Coal Co. has installed several chutes made of "Corros iron." The first trial of these chutes was made about 2 years ago, when a half-dozen sections were installed in the Loree breaker. A typical straight section of one of these chutes is shown in Fig. 22, as well as a typical curved, or spiral, section. This material is cast iron, containing approximately 12 per cent. of silicon. The properties claimed for it are immunity from corrosion by acid water, comparative freedom from oxidation, and extreme resistance to abrasion. The hardness of the metal is fully equal to that of chilled iron, being so great that it cannot be worked by ordinary machine tools.

The chutes installed at Loree breaker, Fig. 23, approximately 2 years ago, show no appreciable wear or deterioration, apart from the fact that the inner surfaces have become smoother and brightened by the passage of material. Installations made more recently have given similar

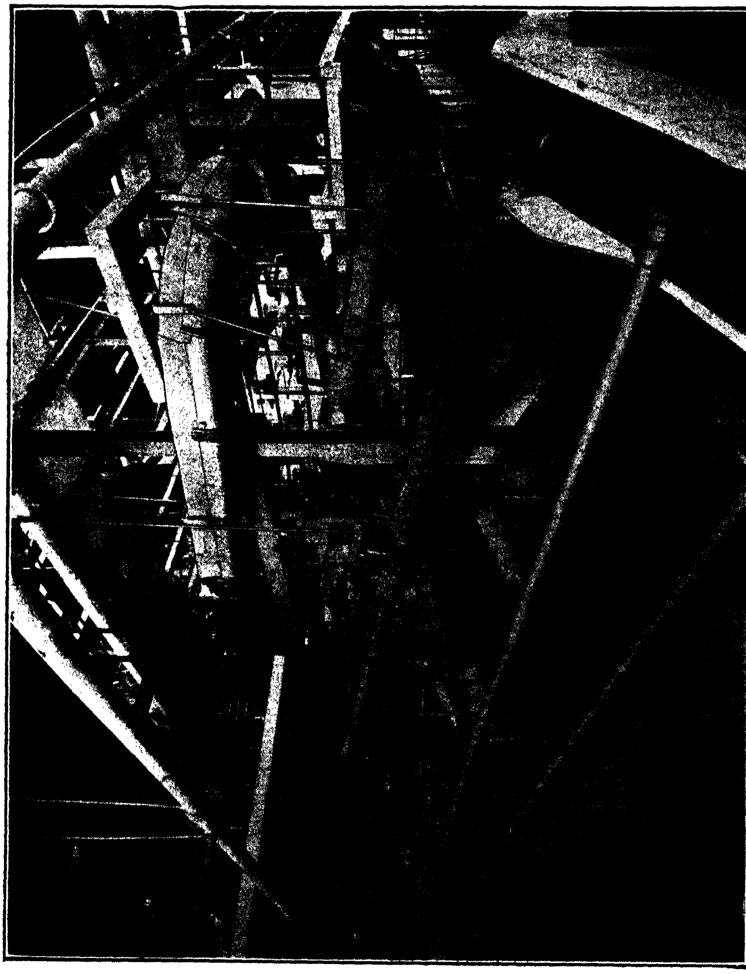

FIG. 23.—INTERIOR OF PART OF LOREE BREAKER OF HUDSON COAL CO., SHOWING A SPIRAL CHUTE MADE OF CORROS METAL.

results. When these chutes are first installed, they have a rough cast finish on the inside, as no practicable method has been devised for making them smooth except that of using extreme care in facing the mold in which they are cast. As a consequence, ordinary pitches are not quite sufficient to carry the material through them when they are first in-

stalled. After they have been in service several months, however, the inside surface becomes smooth and bright and the pitch required is only that ordinarily employed for sheet steel under good conditions. When installing these chutes, it is probably best to place them on such a pitch as is ordinarily required for the operation of smooth-iron or sheet-steel chutes, and line them with iron sheets. Then this lining should be removed, one sheet at a time from the bottom upwards, thus exposing, successively say each day, a new section of Corros iron to the wear of the material passing down the chute.

FIG. 24.—ORIGINAL SPIRAL PICKER USED TO REMOVE SLATE FROM COAL; THE SLATE HUGGED THE INSIDE WHILE THE COAL TENDED TO KEEP TO THE OUTER EDGE. DESIGNED AS A CHUTE FOR LOWERING COAL, ITS VALUE IN SEPARATION WAS ACCIDENTALLY DISCOVERED.

The cost of Corros iron chutes depends largely on the nature of the casting, as the chief expense incurred is that of handling the metal in the foundry. Straight sections of 18-in. (46 cm.) chutes, 3 ft. (91 cm.) long, cost approximately $24 per linear foot. The ordinary breaker chute lined with sheet steel costs approximately $3 per foot. Just how long Corros iron chutes will last has not been determined, but none has as

yet shown appreciable signs of wear. It has been demonstrated that in certain locations Corros iron has already outlasted at least ten renewals of No. 10 sheet-steel chute lining, and bids fair to last indefinitely. The erection of Corros iron chutes requires care, as they are necessarily made in flanged sections of definite size; moreover, this metal is rather brittle, being only about half as strong as ordinary cast iron, so that adequate supports must be provided.

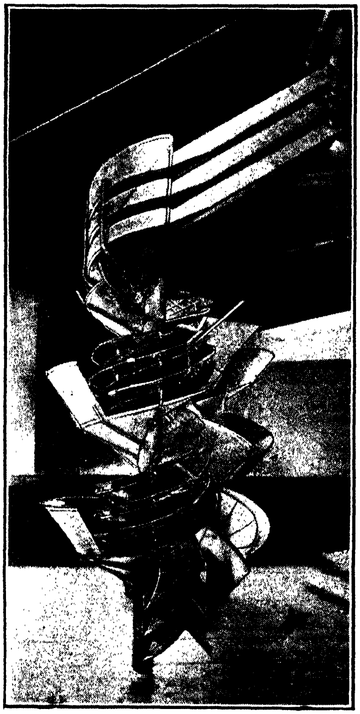

FIG. 25.—MODERN SPIRAL PICKER. THIS CAN BE REGULATED TO MEET ANY AND ALL CONDITIONS IN CHARACTER OF COAL TO BE CLEANED.

SPIRAL PICKERS

The anthracite spiral picker is the one picking device, aside from jigs, that is still in everyday use and shows no sign of disappearing. This contrivance separates the coal from the slate by centrifugal action. The mixed material is fed into the top of the picker and as it descends its velocity increases. As the coal rolls faster than the slate slides, it is thrown into the outside spiral, while slate keeps to the inner spirals and is discharged at the lower end. Fig. 24 shows the first spiral installed; it was built for experimental purposes in 1889. Fig. 25 shows the

modern spiral, which is fully adjustable to any condition that may be encountered. In the latter illustration parts of the sides have been removed to show the inside arrangements.

Tests made last April on different sizes of coal gave the following average results:

	Coal, Per Cent.	Slate, Per Cent.
Material entering spiral	84.60	15.40
Coal leaving spiral	98.93	1.07
Slate leaving spiral	5.88	94.12

The total loss of coal in the slate amounted to 1 per cent. and the slate removed equaled 94 per cent. of that originally contained. Spirals have a capacity of from 8 to 12 tons per hour, depending on the size of the coal.

Fireproofing

The tendency is to build breakers of steel and concrete, thus reducing the fire hazard, though some companies are still building wooden structures. Although all the anthracite companies do not use the same methods of breaker construction, particularly in reference to fireproofing, their ideas are similar. The Hudson Coal Co. has given the following details of its practice at the Marvine breaker. In building new breakers, and adjacent structures, for the past several years this company has taken steps to render such buildings fireproof. The precautions taken to this end may be summarized briefly as follows:

In new breaker structures the framework has been built entirely of steel. As far as possible, the design has provided for complete accessibility to all main members for the purpose of frequent inspections and painting. For protection of the steel against corrosion by acid water, deterioration from rust, and the like, adequate painting with a suitable vehicle (asphalt and carbon chiefly) is relied upon exclusively.

The roofs and side walls, or sheathing, of these buildings have been made of asbestos-protected metal, attached to steel girts and purlins by means of the usual straps and clips. All window openings have been provided with steel sash, glazed with factory-ribbed wire glass, and fitted with top-hinged or pivoted ventilator sections. Practically all door openings have been provided with steel door frames and doors made up of a structural-steel framework covered with asbestos-protected metal.

All floors have been constructed of reinforced-concrete slabs, practically continuous over the entire floor area; these vary in thickness from 4 to 6 in. (10 to 15 cm.), in accordance with expected loads and character of service. In building these floors a self-furring lath, such as Hy-Rib or self-centering, has been laid directly on the floorbeams and fastened in place with suitable clips. The concrete has then been poured on top of this lath, without the use of wooden forms. The underside of the floor is

back-plastered with a cement gun and hand floated to a suitable finish. In addition to the metal lath, small-diameter bars provide further rein-

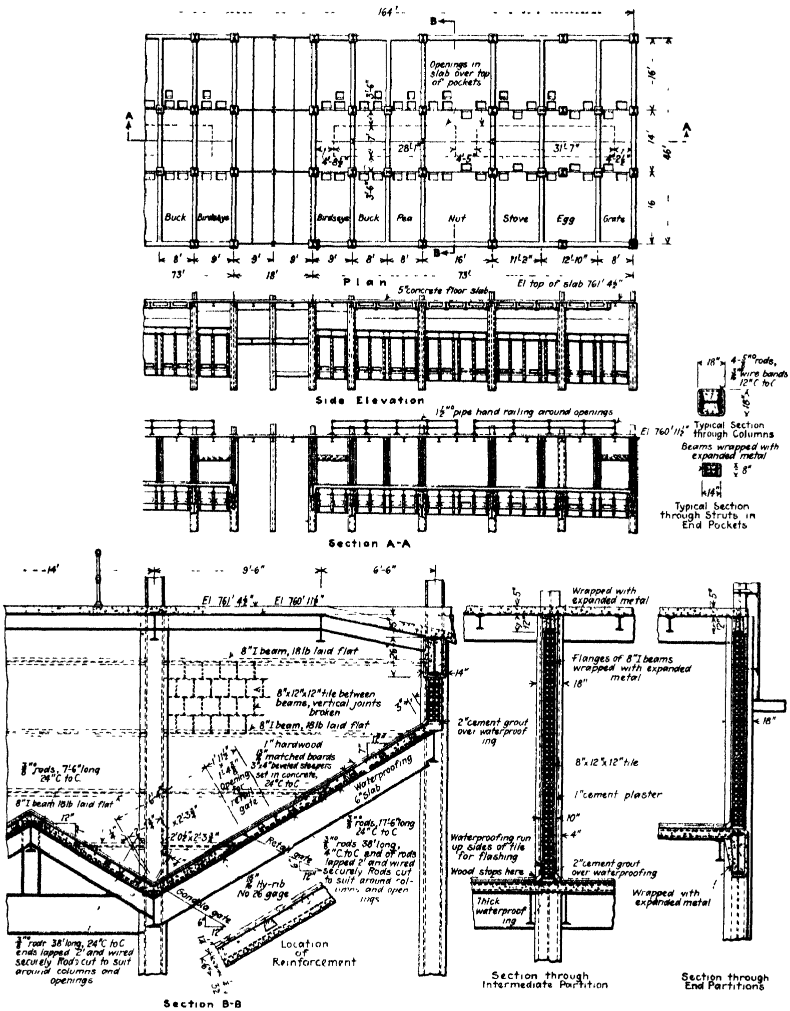

FIG. 26.—SECTION THROUGH POCKETS OF MARVINE BREAKER SHOWING METHOD OF CONSTRUCTION AND MANNER OF FIREPROOFING.

forcement, so that the possible, although unexpected, deterioration of the lath will not necessarily impair the structural strength of the completed floor.

All stairs have been built of structural-steel stringers without risers, and with 2-in. plank treads. The use of wooden stair treads is not thought to add any serious fire risk, and, except several special and expensive forms of treads, it is considered the safest and most satisfactory construction. Pipe hand railing has been used on stairs and throughout the entire structure, except where angle-iron handrails and supports have been thought more suitable.

All loading pockets in the breaker have been built on steel stringers, framing into the steelwork of the breaker. The pocket floor, side walls, and partitions have been constructed of reinforced concrete in practically the same manner as that followed in the erection of the floors, except that hollow tile has been used to some extent for partitions between pockets. This construction is illustrated in Fig. 26. These pockets have been waterproofed by liberally coating the inside surface with an asphalt mastic, and laying in it the wood lining necessary to protect the pocket floors and side walls from abrasion by the coal and the effects of acid water. The lip screens, chutes, hoppers, and troughs have been built entirely of cast iron and steel, the only wood entering into their construction being the gate levers.

Pockets under shaking screens, in the rear of jigs, and the slush troughs under jigs, have been built of almost identically the same construction as the loading pockets of the breaker except that it has not been necessary to build them of equal strength. By the use of concrete floors, pockets, and slush troughs, a continuous monolithic covering is provided over the entire breaker area, level with the tops of the pockets, the only openings being those provided for access by stairs and elevators. This in itself affords obvious advantages from the standpoint of fire-protection, as well as from that of adequately heating the structure.

The use of wood in the newer structures has been confined to the jig tanks, pocket linings, and other points already noted. Heavy planking for the bottom and sides of flight conveyors has not been entirely eliminated, because of the impracticability of fastening the conveyor trough and side plates to any other structural material. In some instances, heavy plank flooring is still used as it has great strength in proportion to its weight and affords a better footing. Furthermore, it is able to withstand vibration and flexure without impairment. It is intended, however, to eliminate entirely the use of wood in this connection.

Fireproofing in connection with electric-motor drives in breakers need cover only the control equipment and the wiring to the motor, the motor proper needing no fireproofed inclosure. The policy followed has been to inclose each oil circuit-breaker in a fireproof cell with all such breakers and control equipment concentrated in one room, which is

made entirely fireproof. The wiring to the motors is incased in a steel conduit for protection against abrasion, and to eliminate the possibility of short circuits between conductors.

The most pronounced step in the direction of securing a fireproof breaker structure is the elimination of all boards and light woodwork, it being a well-recognized fact that heavy timber and planking is ignited with extreme difficulty whereas boards and wood of light weight catch fire with ease.

Certain work has been undertaken in connection with the fireproofing of the older structures, with the idea of eliminating the greater and more obvious risks. The motors, together with their control apparatus, have been housed in small fireproof compartments that provide ample space for the attendant to work in, and for ventilation of the equipment, but preclude the possibility of spreading any fire that might possibly originate in the controlling apparatus, particularly the oil switches. Roofs of breaker buildings, particularly those exposed to sparks and embers from passing locomotives, have been re-covered with asbestos shingles or sheet-asbestos roofing. Practically all permanent additions and repairs to the adjacent structures have been roofed with asbestos material.

Chutes, hoppers, and other exposed woodwork have been fireproofed by applying metal lath, or in some cases chicken wire, and covering this with cement plaster applied either with the cement gun or by hand. Metal lath and gunite are to be preferred to chicken wire and hand plastering In other instances, similar woodwork has been protected by sheets of light-gage steel (No. 24 or 26); this method yields effective and durable results.

Small frame buildings or portions of buildings housing important machinery have been effectively fireproofed by the application of metal lath and cement plaster inside and out, together with the use of asbestos shingles for roofing.

Air Washer

A device for the mechanical cleaning of coal that has recently made its appearance might be termed an air washer or concentrating table, Fig. 27. The basic principle of this machine is simple, and its operation is readily understood. Coal, screened to approximately uniform size, is fed to one corner of an oscillating or jigging table, the bed of which is full of small openings, through which air is forced by a fan built in the base of the machine. The bed is provided with riffles similar to those on an ordinary machine using water as a separating medium and is inclined, both longitudinaly and transversely, both slopes being adjustable within suitable limits. Air pressure, and the length and

rapidity of stroke are also variable. When a sized mine product is fed to this machine, the slate or other impurities settle to the table surface, where they are guided by the riffles, the coal "floating" over the riffles unhindered. The result is that pure coal can be delivered to one point, slate without any coal admixture is delivered to another, while the balance of the material arranges itself between these two extremes in accord with its specific gravity. Products of almost any desired composition and almost endless variety may thus be secured.

One of the great advantages of this machine is that no moisture is added to the coal during treatment. If the coal is already wet from previous processes, such as crushing and screening, an appreciable percentage of such moisture would doubtless be carried away with the air blast. This is of particular advantage in cold weather when wet coal freezes readily after being loaded on to the railroad car.

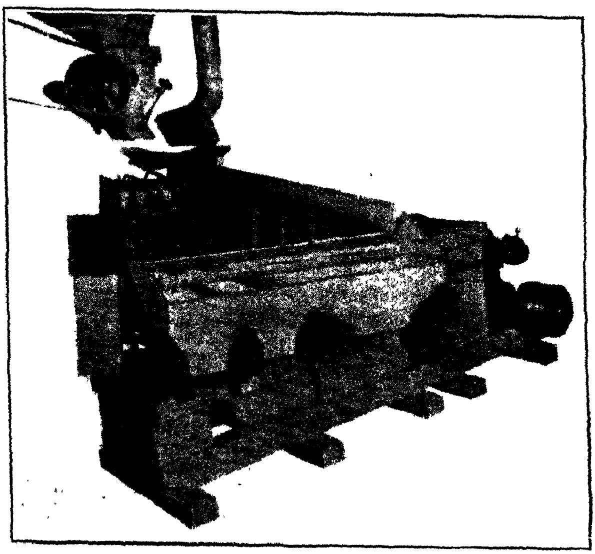

FIG. 27.—AIR TABLE FOR SEPARATION OF SLATE FROM COAL DESIGNED TO TAKE OFF ONLY COAL AND SLATE AND NO INTERMEDIATE PRODUCTS.

As at present built the capacity of this machine varies from about 3 to 10 tons per hour, the quantity that can be successfully handled depending on the size treated, being less with the smaller and more with the larger sizes. The power required to drive the machine is probably about that used in driving an ordinary concentrating table and supplying it with water.

Conklin Separator

The Hudson Coal Co. has recently developed an experimental plant for testing the operation of the Conklin method of separating coal from

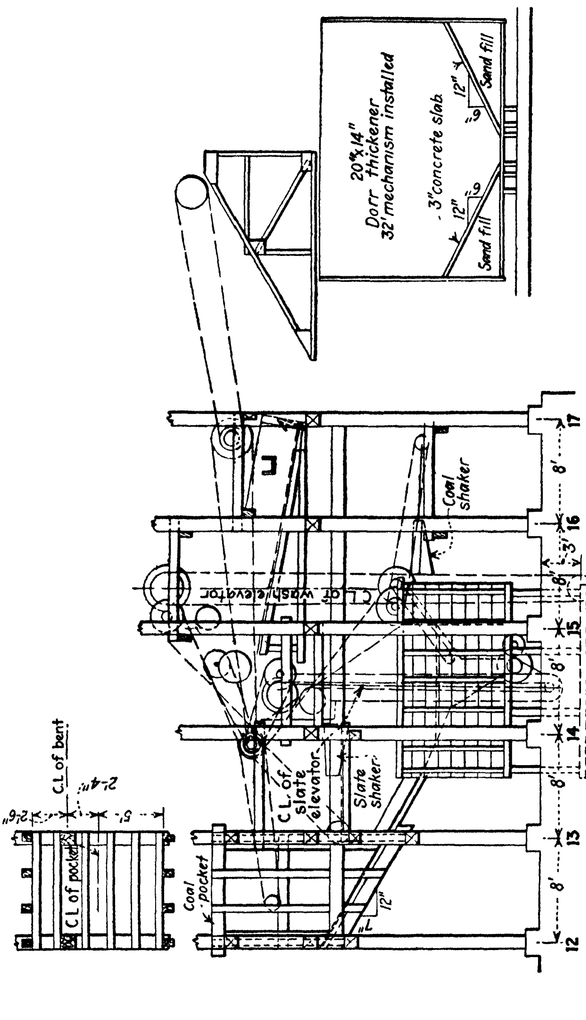

Fig. 28.—Elevation of Conklin separator.

its heavier impurities. This method, or process, is based on the principle of introducing a mixture of coal with its impurities into a fluid medium possessing a specific gravity somewhat greater than that of the coal, and

somewhat less than that of the heavier impurities. The medium used is a pulp composed of magnetite dust passing a 200-mesh screen diluted in the proportion of approximately 44 to 1, and having a specific gravity of about 1.9. This fluid quality is sufficiently permanent to make the pulp a working medium that does not require agitation in order to preserve its property of high density. In fact, agitation of any kind is distinctly harmful to its operation, and is so far as possible eliminated from the process. At this density it is quite fluid and retains this quality to a large extent after 18 to 24 hr. of settlement.

The experimental installation consists of several elements, the first of which is the separator tank, filled with the pulp, into which the stream of impure coal is introduced. The actual separation takes place in this tank, the coal passing off on the surface of the pulp and the slate and other impurities sinking to the bottom and being removed by a screw conveyor and elevator. The removal of the coal is also assisted by a flight conveyor, which elevates it slightly above the overflow point of the tank. At the coal and slate discharges, a small shaking screen, having a very fine mesh, and a water spray are installed to remove any pulp that adheres to the coal and slate. Fig. 28 shows a side elevation of the apparatus.

The second element of the plant is a thickener tank, pump, and Dorr classifier, the function of which is the production of the pulp. The pulp washed off from the coal and slate as these materials emerge from the separator tank is lifted by means of a bucket elevator to a Dorr classifier, which removes the impurities and allows the diluted pulp to flow to a Dorr thickener. In this tank the pulp is restored to its proper degree of dilution, and is removed from the tank by a diaphragm pump which discharges it into the top of the slate elevator, thus completing the circuit.

The apparatus described has not, as yet, been placed in actual operation, but similar equipment, smaller in size, built by Mr. Conklin at Joplin, Mo., indicates that results may be expected approximately as follows: The machine will handle run-of-mine coal with the fines removed approximately as well as it will handle graded and sized material. By fines is here meant buckwheat and smaller, which, at present, it is not thought that the machine will be particularly proficient in handling. The capacity of the separator is expected to be approximately 7 tons per foot of width per hour. By "width" is meant the dimension of the stream at right angles to its direction of flow. With this capacity, separation is secured absolutely in accord with the specific gravity at which the pulp is maintained, so that by slight variations in the pulp density, the quality of the coal and slate can be controlled with a fine degree of nicety. By the use of magnetite-ore dust, it is expected that

such specific gravities of the pulp can be secured that everything except the heaviest pure slate and rock can be supported on its surface.

Dorr Apparatus for Recovery of Anthracite Slush

The problem of most efficiently recovering anthracite slush in as dry a condition as possible has been given much attention, particularly by the Hudson Coal Co., which has found the Dorr hydroseparators and classifiers the most efficient agents for this purpose. An installation at the Loree breaker has given satisfactory results for almost two years. This installation consists of a Dorr hydroseparator 26 ft. in diameter and three Dorr duplex classifiers. The hydroseparator rejects the bulk of the water and the extremely fine solids and slime and gives a partly dewatered product; the classifiers eliminate the remaining fine solids and furnish a more completely dewatered product. The hydroseparator and classifiers also reduce the ash in the slush, as the fine solids and the slime rejected in the overflow are higher in ash than the original total slush.

The hydroseparator is a wooden tank with an overflow launder around its top and a discharge at the center of the bottom, to which are brought the settled solids by means of a rotating plow. The breaker slush, screened to remove all marketable coal, is fed into the top of this tank at the center. The larger particles of coal settle to the bottom and the extremely fine material is carried off in the overflow. The relative sizes of coal recovered and rejected are determined by the settling area of the tank and the amount of material fed to it. At the Loree breaker, the separation is made at 60-mesh, so that most of the plus 60-mesh solids are in the underflow and the minus 60-mesh solids in the overflow. At other installations 34-ft. tanks are planned; these will give a separation at approximately 100-mesh. Material finer than this, as a rule, has over 50 per cent. of ash so that its recovering and purification does not appear economic at the present time.

The underflow from the hydroseparator passes to three Dorr classifiers, each receiving approximately one-third of the total material by means of a three-opening distribution pipe. The classifier consists, essentially, of a settling box or tank in the form of an inclined trough open at the upper end and equipped with mechanically operated reciprocating rakes, which remove the coarse material as it settles to the bottom of the tank, the water and fine solids overflowing at the closed lower end. The solids recovered by the classifiers carry 35 to 40 per cent. of moisture, which soon drains, when stored in an open pile, to between 15 and 18 per cent. moisture.

A summary of the average results obtained in slush recovery by the Dorr apparatus at the Loree breaker is given below. When considering

these results as to percentage recovery of solids, etc., it must be remembered that the plant was designed to save only the plus 60-mesh solids with a minimum of material finer than that size, so that the overall recovery of total solids is necessarily low inasmuch as over 50 per cent. of the total slush is finer than 60-mesh. With a plant designed for finer separation, provided with a larger tank and different classifier adjustment, recovery of practically all the slush solids can be made, but at the sacrifice of ash content in the finished product and with a resultant increase in moisture in the material recovered. It is not economical to recover all the solids, as the extremely fine material and slimes are relatively high in ash, compared to the coarser granular solids, and are also difficult to clean. The elimination of the slimes in the Dorr apparatus is therefore an important step in the effective recovery and utilization of anthracite fines.

A summary of the average results obtained from the Dorr plant at Loree, on anthracite slush, follows:

1. Total solids in slush fed to hydroseparator	49.8 short tons per hr.
2. Average size test on total solids in feed (cumulative):	
plus 60-mesh	44 50 per cent.
plus 100-mesh	57 60 per cent.
plus 200-mesh	72 10 per cent.
3. Average ash content of total solids in feed. .	32 9 per cent.
4. Total solids recovered by hydroseparator and classifier.	22 71 short tons per hr.
5. Average size test on solids recovered (cumulative):	
plus 60-mesh	89 00 per cent.
plus 100-mesh .	97 00 per cent.
plus 200-mesh	99 50 per cent.
6. Average ash content of solids recovered	24 85 per cent.
7. Proportion of total solids recovered by hydroseparator and classifiers .	45 60 per cent.
8. Proportion of total solids lost by hydroseparator	39 10 per cent.
9 Proportion of total solids lost by classifiers	15 30 per cent.
10. Proportion of plus 60-mesh solids recovered by plant .	91 00 per cent.
11 Proportion of plus 100-mesh solids recovered by plant	76 70 per cent.
12. Proportion of plus 200-mesh solids recovered by plant	62 90 per cent.
13. Proportion of total combustible recovered by plant .	51.20 per cent.
14. Proportion of water eliminated by plant .	98 40 per cent.

The plant at Loree at present capacity shows a direct operating cost of 5.3 cents per ton, covering labor, power, supplies and repairs, and a total operating cost of 14.3 cents per ton, including 8.8 cents per ton to cover interest, insurance, taxes, and depreciation at 20 per cent., on an investment of $17,500 for building and equipment. This cost covers delivery of finished product to a conveyor either for stocking or for loading on cars for shipment. The additional cost of operation for this conveyor will, of course, vary with local conditions.

Jigs

The most important apparatus for the cleaning and the washing of coal is the jig. Six important types are used in the anthracite fields though jigs of other types are used in a few places for particuliar purposes. The most important jigs now in use are the Reading, Lehigh Valley, Elmore, Wilmot-simplex, Tench or Delaware, and the James. Two jigs that are used to some extent are the Liberty, or German, and the Christ, which is used mainly for the treatment of the broken size. Only the six first named will be described here.

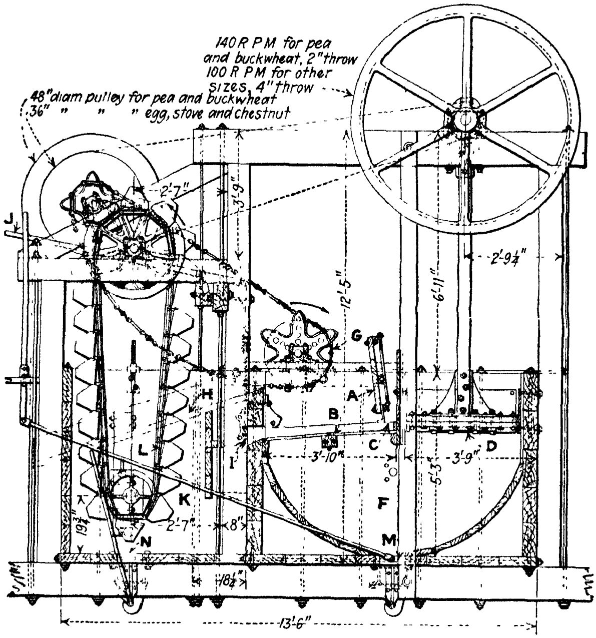

FIG. 29.—SECTION OF READING JIG, USED AT MOST COLLIERIES OF PHILADELPHIA & READING COAL AND IRON CO.

Reading Jig

The Reading jig is of the plunger type and is used mainly by the Philadelphia & Reading Coal and Iron Co. Fig. 29 shows a cross-section of this machine. The coal is fed to the machine in the rear of the baffle plate *A* and passes down behind this plate on to mesh plates *B* through the opening *C*. The water in the jig tub *F* is caused to rise and fall by the plunger *D*. As this water passes through the meshes of the mesh plates, it lifts and drops the coal. As the slate is heavier than the coal, it settles faster and quickly reaches the bottom, the coal remaining at the top. The cleaned coal is removed by the chain *G* to the clean-coal chute

H, whence it goes to the pockets. The slate gate *I*, operated by the lever *J*, is opened from time to time as the experience of the jig operator may dictate. The slate discharges through the slate gate into the chamber *K* from which it is removed by an elevator *L*, which raises and discharges it into the slate chute.

Fine coal broken down during the jigging process passes through the mesh plates into the jig tub, collecting at the bottom whence it is drawn off through the slush gate *M*. In a similar manner, the fine slate that settles in the slate chamber is flushed out by the gate *N*. When treating pea and buckwheat coals, the jig operates at 140 strokes per minute, the plunger stroke being 2 in. (5 cm.). For larger sizes, the speed is cut down to 100 strokes per minute, the length of stroke being 4 inches.

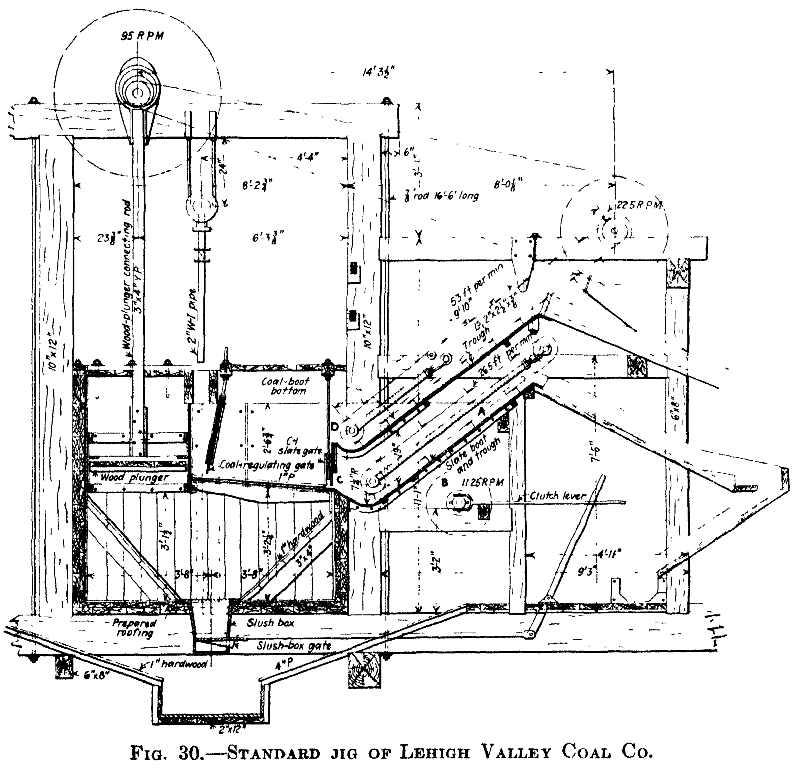

Fig. 30.—Standard jig of Lehigh Valley Coal Co.

Lehigh Valley Jig

The Lehigh Valley jig is quite similar to the Reading. The coal is fed to the machine in the same manner and is treated in the same way but the slate discharge is regulated by the starting and stopping of the slate

conveyor *A*, Fig. 30, by throwing a clutch *B*. This stoppage of the slate conveyor accomplishes the same purpose as the shutting of the slate gate on the Reading jig, for when the conveyor stops there is no escape for the slate and the bed of this material grows thicker on the mesh plates, whereas in the Reading jig it can continue to fall into the slate compartment.

This jig is really an overflow-type machine. No mechanical means is provided for removing the coal from the jigging chamber; it must flow off into the coal boot *D*, from which it is removed by a conveyor. The slush is removed from the bottom of the jig tub through an aperture that can be opened and closed at will.

The accompanying table shows the results obtained from the use of these jigs in both the upper and the lower coal fields. These results are interesting not only from the point of view of the performance of the Lehigh Valley jig but also because they contrast the conditions encountered in the upper and the lower coal fields.

Upper Field

Size of Coal	Coal Before Jigging, Per Cent. of			Coal After Jigging, Per Cent. of			Slate After Jigging, Per Cent. of	
	Coal	Slate	Bone	Coal	Slate	Bone	Coal	Slate
Egg	81	14	5	97	1	2	1	99
Stove	80	16	4	94	2	4	1	99
Nut	80	15	5	94½	2½	3	½	99½
Pea	84	12	4	91	6	3	½	99½
Buckwheat	83	12	5	84	8	8	1	99

Lower Field

Size of Coal	Coal	Slate	Bone	Coal	Slate	Bone	Coal	Slate
Egg	69	28	3	97	1	2	2	98
Stove	70	27	3	94	3	3	3	97
Nut	74	22	4	93½	3½	3	2	98
Pea	76	20	4	90	7	3	4	96
Buckwheat	76	20	4	87	10	3	4	96

Elmore Jig

The Elmore jig, shown in Fig. 31, also causes the pulsation of the water by means of a plunger. The difference between this and the preceding jigs, however, lies in the method employed to discharge the slate and coal. The coal is withdrawn by means of a direct overflow instead of being assisted by drag-line conveyors or scrapers. The slate discharge is automatic and when set to provide for a certain amount of rock in the coal it requires no further attention. When the weight of the slate in the bed reaches a certain point, it acts against a lever which, in turn, allows the slate to discharge into a small slate pocket, from which it is removed by a drag-line conveyor discharging into a slate chute.

This type of jig requires about 700 gal. per min. of circulating water; 12 hp. are required to operate the jig and 18 hp. to drive the circulating

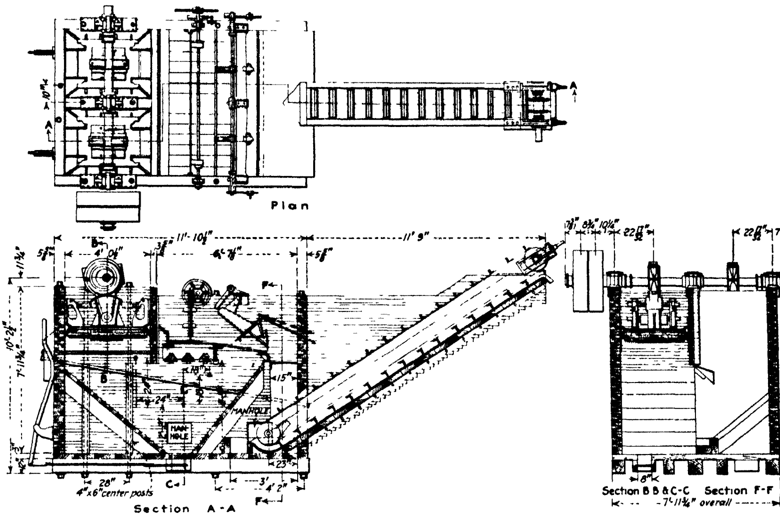

FIG. 31.—DETAILS OF ELMORE JIG.

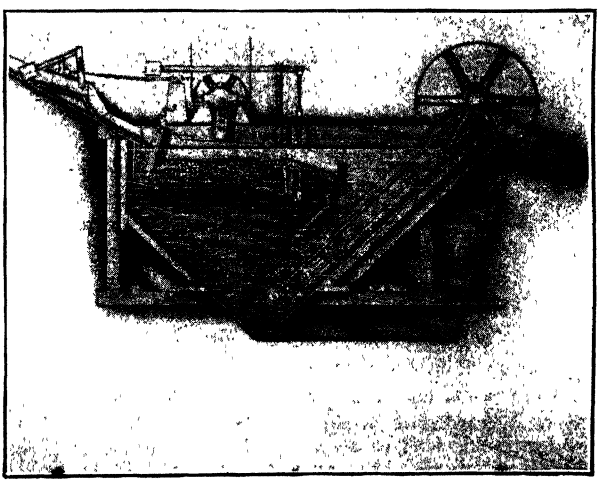

FIG. 32.—WILMOT-SIMPLEX JIG, WHICH IS OF THE PAN TYPE.

pump. It requires a floor space of 24 by 10 by 11 ft. (7.3 by 3 by 3.3 m.). Its capacity is as high as 50 tons per hour. It is in reality, however, a double unit having two jigging compartments and two plungers.

Wilmot-simplex Jig

In the Wilmot-simplex jig, shown in Fig. 32, no plunger is used. The pan containing the coal to be cleaned is moved up and down in the tank; as the water passes through the openings in the pan, it agitates the contents of the pan and the slate settles at the bottom. The coal and slate are both discharged, under water, into semi-pockets, conveyors dragging them out and emptying them into appropriate storage bins.

The jig works automatically, ceasing to run when it is supplied with insufficient coal for its operation. This regulation is accomplished by a pan in the feed chute. When the bed of the jig is properly filled, the coal backs up in this chute until the pan is weighted down, whereupon the jig commences operating. When sufficient coal to weigh down the pan is not present, the pan rises and the jig is shut off. Slush is removed from the jig tub in a manner similar to that already described.

As this jig is widely used throughout the anthracite field, tests made in various places will be of interest. At the Thomas colliery, trials were made on the jigging of a rock that contained a little coal. In the first test, the material going to the jig contained 80 per cent. of rock and 20 per cent. coal. The refuse coming from the jig showed only 0.3 per cent. coal, 0.5 per cent. bone, and 99.2 per cent. slate. In a second test, run on material of the same character and analysis, the same results were obtained in the refuse, the coal containing 10 per cent. slate and 16 per cent. bone. A third test on the same material, made a day later, showed 0.4 per cent. coal and 0.5 per cent. bone in the slate. On the following day, four more tests were run showing an average of 0.3 per cent. coal and 0.3 per cent. bone in the slate. Results secured in two tests on the coal showed that, in the first instance, 14 per cent. slate and 18 per cent. bone were present; in the second test, there was 10 per cent. slate and 12 per cent. bone in the coal. A fifth series of tests was made on the same material the following day, and the same results were obtained for the slate; the coal showed 10 per cent. slate and 10 per cent. bone

In the Enterprise Coal Co. tests on the type *D* simplex jig, using stove size containing 30 per cent. coal and 70 per cent. slate, and chestnut containing 50 per cent. coal and 50 per cent. slate, in the first test, and a mixture of 25 per cent. coal and 75 per cent. slate in the second test, the following results were obtained:

Stove slate contained ½ per cent. coal.
Chestnut slate contained ½ per cent. coal.
Stove coal contained 2 per cent. slate and 1½ per cent. bone.
Chestnut coal contained 3 per cent. slate and 3 per cent. bone.

At a Lehigh Valley Coal Co. colliery in the lower field, tests made on material that contained 50 per cent. coal, 44 per cent. slate, and 6 per

cent. bone gave the following results: The coal contained 95¼ per cent. coal, 2¼ per cent. slate, and 2½ per cent. bone; and the refuse contained 3¾ per cent. coal and 96¼ per cent. slate.

Delaware, or Tench, Jig

The Delaware, or Tench, jig, an installation of which has been made by the Hudson Coal Co., is a modification of the Lehigh Valley jig. A lifting plunger takes the place of the coal conveyor and a conveyor line

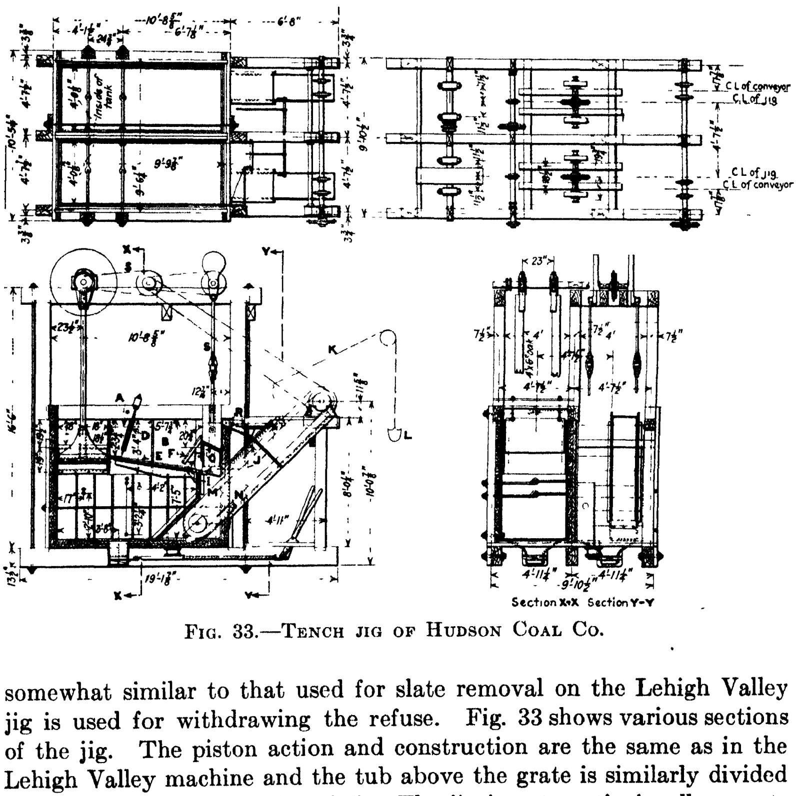

FIG. 33.—TENCH JIG OF HUDSON COAL CO.

somewhat similar to that used for slate removal on the Lehigh Valley jig is used for withdrawing the refuse. Fig. 33 shows various sections of the jig. The piston action and construction are the same as in the Lehigh Valley machine and the tub above the grate is similarly divided into two unequal parts *A* and *B*. The jig is automatic in all respects and there is no need of discharge floats or power-driven slate-discharge devices. The coal and slate are fed into the compartment *A* by a chute and pass underneath the feed-regulating gate *D* on to grate *E*, where they form a bed composed of coal and its impurities. The action of the piston effects a separation in the same manner as in any other piston type of jig.

As the bed of coal and slate forms upon the jig grate *E*, the slate comes into contact with the perforated guard *F*, which extends across

the front of the jig and reaches to within 4 in. (10 cm.) of the jig grate *E*. Part of the slate passing under this guard *F* comes into contact with the slate gate *I* and exerts pressure against it. This gate is attached, by rods *J*, to a rope *K* passing over a pulley, from the end of which is sus-

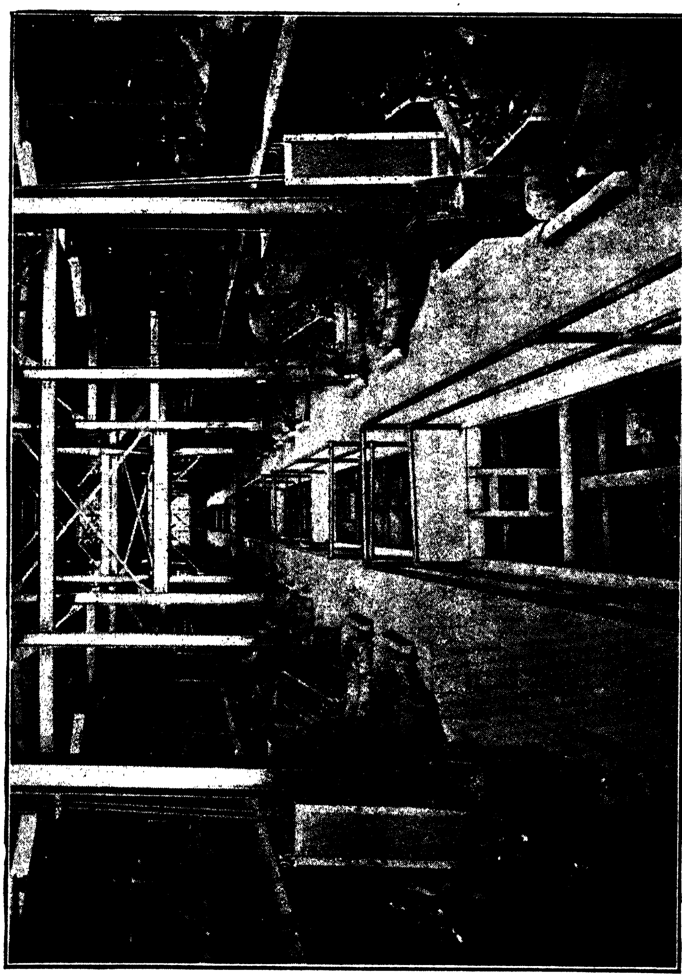

FIG. 34.—INTERIOR OF MARVINE BREAKER SHOWING JIG FLOOR.

pended a small bucket *L*; weights placed in this bucket control the amount of slate discharged from the jig. The weight to be used must be determined by experiment for the different sizes and grades of slate being jigged. When the pressure of the slate against the gate becomes greater than the force exerted by the weights in the bucket *L*, the gate opens.

The slate then passes into compartment M, from which it is removed by the conveyor line N. When the pressure against the gate I is decreased so that the force of gravity on the bucket L becomes greater than the slate pressure against it, the gate closes and retains the slate until such time as its pressure again exceeds that exerted by the weights in the bucket L.

As the coal on the top of the bed is floated off, it slides down over the top of the perforated guard F on to the lifting plunger O, which raises it to the top of the jig tub and out of the water. It then slides into the chute R, which leads to the coal pockets. The coal-lifting plunger O is operated by the mechanism S, being so regulated that one stroke of this lifting plunger is made for every four strokes of the jig piston.

The coal-lifting plunger and the slate control on the jig gates are the two distinctive details in the construction of this jig, but a feature that has proved highly beneficial in the removal of flat slate has been the increase in the height of the total jigging bed at the discharge end from 16 to 21 in. (40.6 to 53 cm.). Another advantage is the increase of the grate area, which is approximately 25 per cent. greater than the common size of Lehigh Valley jig.

The coal-lifting plunger O is made of cast iron, the side plates forming the end guides and the front and rear plates forming the side guides. Removable perforated plates bolted on to the slanting top surface let the water and fine material return to the jig tub; the perforations in these plates must be suited to the size of the coal being jigged. The back, or long, surface of the coal-lifting plunger also is equipped with perforated plates, which insures a free passage of the water between the main jig tub and the plunger compartment and lessens the splash of the water when the plunger makes its return stroke. It also forms a back plate to the plunger and prevents the coal in compartment B from falling under the plunger O into the tank below. The force of the water passing through its perforations holds back the coal.

When the plunger O is in its lowest position, its uppermost point falls slightly below the lower edge of the slanting surface of the inclined perforated guard F. The coal on the top of the bed in compartment B, because of the action of the jig piston, is thus thrown on to the top slanting surface of the coal-lifting plunger.

The slanting surface of the plunger O is inclined toward the front of the jig at an angle of about 30°; this allows the coal to slide off into chute R when the plunger is at the highest point of its movement. As soon as the coal on the plunger extends above chute R sufficiently to let it seek its angle of repose, it enters this chute so that when the plunger has reached its full stroke, little or no coal remains on its surface.

The advantages of this jig are: (1) Automatic slate discharge, (2) only one small conveyor line, (3) slight repairs, (4) all water remains in

the jig, (5) little coal is broken, (6) less space is required for a given capacity, (7) cleaner separation is secured, (8) less water is required than in the overflow type of jig. This jig is now in successful operation on all sizes of coal from broken or grate to No. 1 buckwheat, inclusive. The following is a test on stove coal treated in this jig. Every ten minutes samples were taken of the feed, clean coal, and slate.

Material Entering Jig from Twenty-four Samples, Aggregating 259.25 Lb.

Size of Material	Size, Per Cent.	Coal, Per Cent	Bone, Per Cent.	Slate, Per Cent.
Stove	93 3	68.9	2.2	28.9
Nut	6 2	81.3		18 7
Pea	0 2	100.0		-
Smaller	0.3			

Material Leaving Coal End of Jig From Twenty-four Samples Aggregating 239 Lb.

Size of Material	Size, Per Cent.	Coal, Per Cent	Bone, Per Cent	Slate, Per Cent
Stove	90.6	96 5	1 4	2 1
Nut	9.1	100.0		
Pea	0 2	100 0		
Smaller	0.1			

Material Leaving the Slate End of Jig from Twenty-four Samples Aggregating 264.25 Lb.

Size of Material	Size, Per Cent.	Coal, Per Cent.	Bone, Per Cent.	Slate, Per Cent
Stove	88.6	1.4	0.8	97.8
Nut	9.4	3 0		97 0
Pea	1.4	33.7		66 7
Smaller	0 6			

The pea and smaller sizes coming from the slate end are screened out before they reach the main slate conveyor, and are sent through the breaker again.

Breakage During Test

	Per Cent
Stove size entering the jig	93.3
Stove size leaving the jig	89.6
Breakage due to the jig	3.7

This breakage includes chestnut, pea, and smaller sizes.

James Jig

The James jig is of the single-compartment, balanced type, using the cup-and-gate method of refuse discharge. The screen is provided with 1/4-in. (6 mm.) circular perforations and carries a bed of 3/8-in. iron punchings.

Fig. 35 is a cross-sectional view. The device consists of a stationary jig chamber, supported in a large wooden tank, on top of which is placed the operating mechanism. From the bell cranks, 1 and 2, the pulsator A is suspended by rods working clear of the sides of the jig chamber. This pulsator has the shape of two inverted pyramids, on

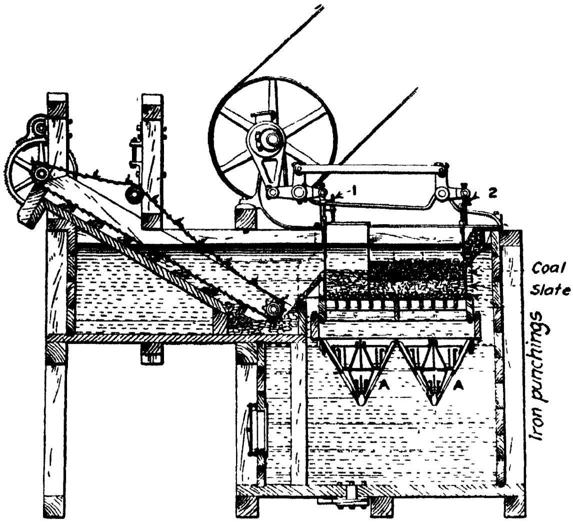

FIG. 35.—SECTION OF JAMES JIG.

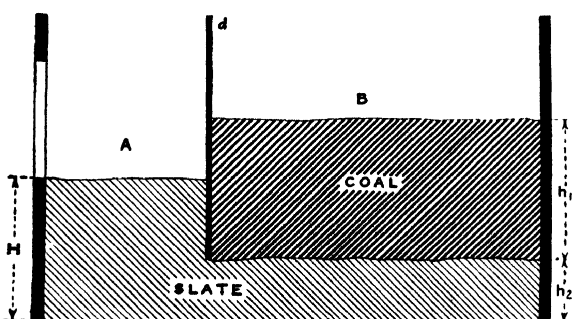

FIG. 36.—DIAGRAM SHOWING METHOD OF REMOVAL OF SLATE FROM JAMES JIG.

each face of which are placed three valves, making twenty-four in all, having an aggregate area equal to that of the screen. These valves are so designed that they open on the downward and close on the upward stroke of the pulsator, thus causing the flow or pulsation of water through the jig to take place in one direction; this reduces suction to a minimum.

Coal is fed to the jig by a chute leading in on the right, and after separation it overflows through an opening on each side of the chamber, as shown by dotted lines. It is discharged by drag lines, not shown. The slate discharges on the left side of the jig chamber and feeds the drag line illustrated.

Fig. 36 shows that the jig chamber is divided by the baffle *d* into two parts *A* and *B*; this baffle clears the top of the screen by a distance equal to the thickness of the bed desired. Water pulsates through the bottom screen in both compartments *A* and *B*. The coal and slate are fed into the right end of the chamber *B*. When compartment *B* has a certain height, or head, of slate h_2 and coal h_1, the water level being the same in both compartments (*A* and *B*), the sum of the weights of these heads must be equal to the weight head of the slate in compartment *A* at any given instant. If a means of discharging the coal in compartment *B* and the slate in compartment *A* is provided, each upward pulsation of water will cause coal to overflow in *B* and slate in *A*, when and so long as $W_2H = w_1h_1 + w_2h_2$, where w_1 and w_2 are the specific gravities of coal and slate, respectively.

The height of the two discharge gates, of course, takes into consideration the average specific gravity of the materials to be separated. A numerical example will show this action. Assume the specific gravity of the slate to be 3 and of the coal 1.5, and that the bottom of the baffle *d* is set 3 in. (7.6 cm.) above the screen, the coal discharge gate being set 15 in. (38 cm.) from the screen.

The head due to the slate is 3 in. $\times$ 3 = 9 units.

The head due to the coal is (15−3) in. $\times$ 1.5 = 18 units.

Total head = 18 + 9 = 27 units.

This figure divided by 3, the specific gravity of slate, gives the height of the slate discharge *H* above the screen, or 27 ÷ 3 = 9 in.

The height of the coal overflow is fixed, whereas that of the slate can be varied to suit the material treated. Setting the slate overflow, or gate, too low will cause a rich refuse, and placing it too high will cause slate to come over with the coal

In Fig. 35 the division plate *d* is shown at the left of the jig chamber. The crank handle to the left controls the height of the slate overflow, and the rate of feed to the jig is regulated by the handle to the right, operating a gate on the feed chute. The coal overflow is on both sides of the chamber; it is represented by dotted lines.

The following advantages result from this type of construction: (1) Low water consumption; (2) automatic operation, for no "tapping" is necessary; (3) operation and efficiency independent of the refuse content of the coal, up to the maximum rate at which the jig will discharge refuse; (4) practical elimination of suction; (5) equally efficient operation with anthracite from ½ in. (12.7 mm.) to 1/16 in. (1.6 mm.); (6)

consistent results in anthracite operation; the ash content of the coal overflow has not varied more than 1 to 2 per cent. during 10 months of operation.

Tests on the James jig at the plant of the Locust Mountain Coal Co. gave the following results:

Feed	Coal, Per Cent.	Slate, Per Cent.	Ash, Per Cent.
13.3 Tons of Feed per Hour			
Feed	78.5	21.5	20.13
Slate as discharged	8.0	92.0	
Coal as discharged	90.5	9.5	13.0
14 Tons of Feed per Hour			
Feed	80.5	19.5	20.35
Slate as discharged	7.5	92.5	
Coal as discharged	91.5	9.0	13.50
19.3 Tons of Feed per Hour			
Feed	72.5	27.5	24.00
Slate as discharged	4.0	96.0	
Coal as discharged	83.0	17.0	17.13

Chance Sand-flotation Process

Any two lump substances of different specific gravities may be effectively separated by introduction into a liquid, the specific gravity of which lies between their specific gravities. All the particles heavier than the liquid will sink while those lighter will float. H. M. Chance procures any desired specific gravity of the separating liquid by the addition of fine sand to water; this sand is kept in suspension by agitation. For the separation of coal from slate and bone, ocean-beach sand has been used in sizes ranging from 20 to 30-mesh down to 100 to 200 mesh and even finer. Specific gravities of from 1.20 to 1.75 may be maintained for any period.

The inverted-cone type of washer has been used in the most recent of these experiments, the washed coal and refuse both being removed from the apparatus without the use of complex devices or conveyors. A slow-moving rotary stirrer within the cone will keep the sand agitated and prevent its forming into banks on the walls. This fine granular material virtually forms a stratum of quicksand in the lower half of the cone which the stirrer maintains at a uniform density. As the flow of water is reduced to a minimum, a high fluid density is maintained. The cleaned coal is usually discharged through an overflow weir along with the water, but in some cases it is removed with a conveyor or a raking wheel. The coal is discharged on to a stationary screen, where the sand particles that adhere are rinsed off and the coal recovered. In treating the finer sizes, a shaker screen will probably be more efficient.

In washing anthracite, it has been found possible to produce washed coal carrying no free slate and only such a proportion of bone coal as is desirable in the finished product; also, to produce refuse with no free coal of the average ash content of the washed coal. Occasionally, pieces of what appear to be pure coal are found in the refuse but these are invariably found to be exceedingly heavy.

If the average specific gravity of the coal to be washed is 1.5 and the average density of the ash is such as to produce an increase in density of 0.01 per cent. for each per cent. of ash content, a specific gravity of the fluid of 1.6 will produce washed coal, no piece of which can contain more than 10 per cent. of ash. The coal that floats is a high-grade product. The material that sinks can be passed to a second washer, in which the fluid mass is maintained at a specific gravity slightly higher than the first and graded into middlings and tailings. The middlings will contain most of the bone, which can be crushed so as to separate the coal and the rock; it then can be returned to the first washer for cleaning. Pyrite can be separated from the tailings by a third washer and sold as a byproduct.

Retreatment of the tailings from the first washer will be of greater economic value in the case of an efficient cleaning process than with the usual processes. One fact that militates against the production of high-grade washed coal by ordinary methods is the presence of laminated slate and bony coal. In the center of the jig, such material will often have a falling velocity practically equal to that of the large pieces of clean coal and will, therefore, be discharged with the washed product. With the Chance process, no difficulty has been found in maintaining such a fluid density that no individual piece of coal is discharged that contains more than 3 per cent. of pyritic sulfur.

The first commercial installation will be located in the Pennsylvania anthracite district. Equipment is now being designed for the plant, which will be used for the cleaning of practically unsized coal. It has been found entirely possible to concentrate anthracite averaging 20-mesh, and experimental work has been done on coal of smaller dimensions. It is possible to treat an unsized product carrying a large percentage of fine coal, because the fines are discharged with the supernatant wash water at the top of the fluid mass. When extremely small sizes of coal are treated, it is not practicable to screen out all of the fine sand that is removed from the apparatus with the washed coal; consequently hydraulic classification will be used for separating the very fine coal from the sand that is withdrawn with it.

Highly satisfactory results have been obtained in treating No. 1 buckwheat, rice, and barley coals. It has been possible to reduce the impurity so that practically only the inherent ash remains. As a commercial proposition, however, this would result in too great a rejection

of bony coal and hence in too low a recovery. As a result, the following percentages have usually been found to represent the best practice.

	Feed, Per Cent.	Washed Coal, Per Cent.	Reject, Per Cent
Ash	38 00	11 22	83.58
Total weight	100 00	63.00	37 00

Little sand is lost in the operation. When rice coal has traveled less than 1 ft. (305 mm.) over a ⅛-in. (3.6 mm.) mesh screen, the washed coal contains less than 0.6 per cent. of residual sand. A further travel of 1 ft., with the addition of fresh water, reduces this final sand content to approximately 0.1 per cent. or 2 lb. per ton of coal. The sand is washed from the coal by the agitation water after it is discharged over the weir at the top of the cone. It is possible to use this water several times, by employing a screen built in a number of steps, the sand washed out in one portion being given an opportunity to settle before the water is used in the next.

Concentrating Tables

After investigating five methods of cleaning anthracite slush, either on a commercial scale or in the laboratory, the Hudson Coal Co. has obtained the best results with the Deister-Overstrom, diagonal-deck No. 7 coal-washing table. The concentrating table, Fig. 37, has found wide application for years in the metal-mining industry and later for washing bituminous coal. It has only recently invaded the anthracite field, but about 150 of these tables are already in operation or are being constructed for use in cleaning various sizes of anthracite from No. 1 buckwheat down to slush.

The Hudson Coal Co. has made a long series of experiments on anthracite slush at the Loree breaker, Fig. 38, in conjunction with the Dorr slush-recovery plant. The results were highly satisfactory as to ash and sulfur reduction by the washing process, but a table will effectively clean only 3 to 4 tons of slush per hour; though of the larger sizes, up to No. 1 buckwheat, as much as 10 to 12 tons per hour can be handled. With slush, the fineness of the particles precludes the treatment of more than 4 tons, which means that a comparatively large installation (eight tables) is required to treat the slush from the company's largest collieries.

The various operations are as follows: Raw coal, mixed with about twice its weight of water, is delivered to the table through the feed box at the upper corner at the head-motion end of the deck. Water-distributing boards placed along the same side of the deck as the feed box allow a nice adjustment in the distribution of water over the deck surface. The table is placed in a horizontal position and is practically level longitudinally, or along the line of its reciprocation. A slight side inclination

at right angles to this line, adjustable to meet changing conditions, permits the clean coal to be washed over the long edge of the table into a trough or launder. Simultaneously the action of the head motion in reciprocating the deck approximately 275 times per minute with a length

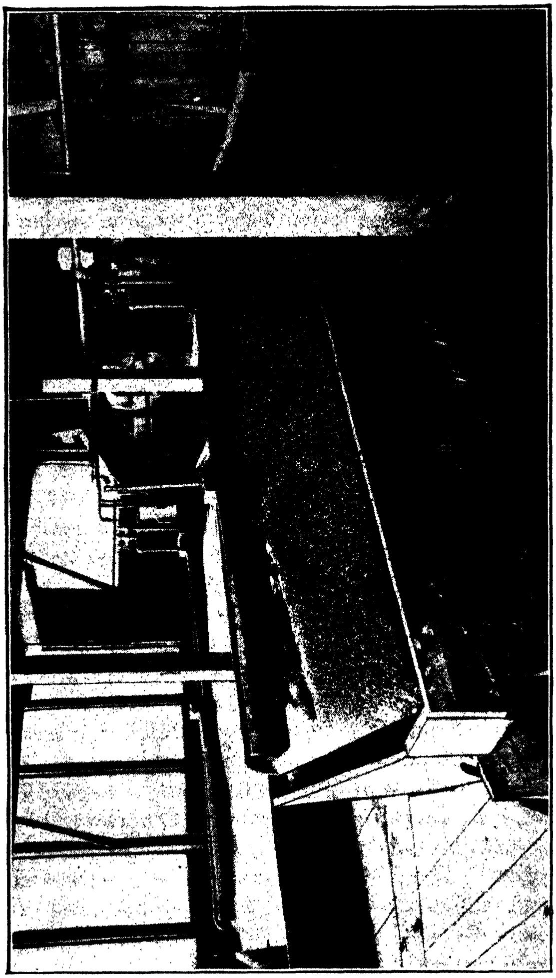

FIG. 37.—TWO DEISTER TABLES FOR PREPARATION OF RICE COAL IN A BREAKER OF PHILADELPHIA & READING COAL AND IRON CO.

of stroke of about ¾ in. (19 mm.), drives the pyrite and refuse, which stratifies next to the surface of the table deck, over the short edge, or refuse end, of the table, where it is caught in launders and conveyed to the refuse heap. The wooden riffles on the surface of the table deck aid in collecting and guiding the refuse to its proper point of discharge

FIG. 38.—LOREE BREAKER OF HUDSON COAL CO. IN SMALL BUILDING AT END OF CONVEYOR LINE IN LEFT FOREGROUND DORR SEPARATOR IS HOUSED.

and also prevent the finer particles of waste matter from washing over with the clean coal.

Using this table on anthracite slush, the Hudson Coal Co. has obtained an average ash content of 13 per cent. in the washed product when the crude coal averaged 28 per cent. ash. This ash content in the washed coal can be reduced still further, to from 7 to 8 per cent. if required, by extreme care in tabling and at a considerable sacrifice of capacity. Typical average results obtained in the operation of this table are as follows:

The material treated on table was anthracite slush [through $\frac{3}{64}$-in. (1.2 mm.) round opening] recovered from breaker slush by Dorr hydro-separator and classifiers, with the following average size as determined by test: On 60-mesh, 88 per cent.; on 100-mesh, 8 per cent.; on 200-mesh, 3 per cent.; through 200-mesh, 1 per cent.

				Distribution and Yields			
	Short Tons per Hour	Ash, per Cent	Sulfur, per Cent	Total Solids, per Cent	Combustible, per Cent	Ash, per Cent	Sulfur, per Cent
Feed.	3.41	28.0	1.71	100.0	100.0	100.0	100.0
Washed coal	2.43	13.0	0.79	71.3	86.1	33.2	32.8
Slate, etc	0.93	65.0	1.91	27.2	13.2	63.2	33.2
Pyrites	0.05	70.0	42.00	1.5	0.7	3.6	36.8

Reduction of ash, 53.51 per cent.; reduction of sulfur, 53.8 per cent.

The best data at hand indicate that 4c. per ton will cover labor and power expense of tabling and that probably the total cost per ton, including fixed charges, depreciation, repairs, etc., will not exceed 10c. per ton in a fair-sized installation, which is one of about eight tables. In this calculation, power is charged at 1.5c. per kw.-hr. and labor at 50c. per hour.

At one of the breakers belonging to the Madeira-Hill & Co. interests, a mixture of rice and barley was washed on Deister-Overstrom concentrating tables for 10 days, the float-and-sink being used to determine the quantity of slate in the washed coal. During the period that the tests were made on the concentrators, 3,600 tons of rice and barley coal were shipped. Samples were taken at intervals of 20 min. and then mixed and quartered to form the final sample. The results were as follows:

Coal Discharged from Concentrator

	Coal, per Cent.	Slate, per Cent
Rice over $\frac{3}{16}$-in. mesh..	87.5	12.5
Barley over $\frac{3}{32}$-in. mesh	87.0	13.0
Barley over $\frac{1}{16}$-in. mesh	87.5	12.5

Slate Discharged from Concentrator

Proportion of slate....	90.75
Proportion of coal..............................	9.25

The material fed to the concentrator contained 30 to 35 per cent. ash. The following is an analysis of the product from the concentrator:

COAL DISCHARGED FROM CONCENTRATOR

	RICE OVER 3/16-IN.	BARLEY OVER 3/32-IN	BARLEY OVER 1/16-IN.
Fixed carbon, per cent	75.93	79.85	80.11
Ash, per cent	19.48	15.20	14.62
Heat value (dry basis), B.t.u.	12,172	12,573	12,663

SLATE DISCHARGED FORM CONCENTRATOR

Fixed carbon, per cent	22.69
Ash, per cent	70.20
Heat value (dry basis) B. t. u	3,636

WATER USED FOR WET PREPARATION

An abundance of water must be supplied for the wet preparation of coal. Where coal is prepared, in general, by dry methods, water is used for the preparation of the finer sizes. The average quantity of water used in the Wyoming Valley per ton of daily output, in preparation by wet methods, is 1,035 gal. (3.9 li.) per min. Thus, if a breaker has an output of 1,000 tons per day it will require 1.035 gal. of water per minute. A combination wet-and-dry method requires 0.634 gal. (2.4 li.) per min. per ton of daily output. The dry method requires only 0.626 gal. per minute.

In the Lehigh region, the wet method of preparation requires 1.428 gal. per min. of water per ton of daily output, and the combination wet-and-dry method of preparation requires 0.692 gal. per min. per ton of output per day. In the lower field, the wet method requires 1.542 gal. per ton of daily output and the wet-and-dry combination method requires 1.23 gal. per ton of output per day. The amount of water required depends on mining conditions. The steep-pitching measures produce a coal that demands more water in its cleaning than does the flat-measure coal produced in the Wyoming Valley field.

Not only is much water necessary, but its quality is important. Ordinary mine water is much too high in sulfur to be employed, as it will corrode the lining of the chutes excessively. Consequently, the coal companies endeavor to obtain either pure water or such as is contaminated with as little sulfuric acid as possible. One company treats its water with quicklime to counteract the sulfur, thus reducing the acidity and prolonging the life of its chutes; it requires about 2 to 4 lb. of lime to counteract the sulfur in 1,000 gal. of water, under average conditions.

FORCE EMPLOYED AT PREPARATION PLANTS

The following figures show the number of tons prepared per man employed in preparation and the relationship between the number of

employees in the preparation plants and those engaged in other portions of the operations:

WYOMING VALLEY FIELD

	Preparation Method Employed		
	Dry	Wet-and-dry	Wet
Number of collieries reporting	14	22	30
Tons of coal produced	5,674,010	10,845,542	17,120,602
Tons of coal prepared per man employed on preparation	7,020	6,420	7,120
Percentage relation of preparation men to outside employees	38.7	34.4	34.3
Percentage relation of outside employees to total employees	20 8	24.1	22 2
Percentage relation of outside employees to inside employees	24.8	31.6	28.6

LEHIGH FIELD

Number of collieries reporting	5	4
Tons of coal produced	2,241,783	1,597,216
Tons of coal prepared per man employed in preparation	4,930	6,000
Percentage relation of preparation men to outside employees	27.9	25 6
Percentage relation of outside employees to total employees	38.6	36 4
Percentage relation of outside employees to inside employees	68.0	57.2

SOUTHERN FIELD

Number of collieries reporting	7	64
Tons of coal produced	2,655,615	20,352,232
Tons of coal prepared per man employed in preparation	5,290	5,730
Percentage relation of preparation men to outside employees	33.5	26.0
Percentage relation of outside employees to total employees	31.2	34.0
Percentage relation of outside employees to inside employees	44.7	50.9

These figures show that more coal is produced per man in wet methods of preparation than in the combination wet-and-dry; this is to be expected for the wet-and-dry method practically means two breakers in one, and a greater force of men will necessarily be required in such a case. The figures also show that in the Wyoming field almost as much coal is produced by the dry method as by the wet method. If the figures here presented are taken without analysis, it will appear that no great saving in men per ton prepared is attained by the use of wet-preparation methods. It should be stated, however, that in the few collieries where

dry methods are being used, a large proportion of the coal is coming from virgin territory and consequently has but little rock mixed with it, so that the force of men required to eliminate the rock is correspondingly reduced. The tonnage produced per man in the upper field is greater than in the Lehigh or lower regions; this is because of the mining conditions prevailing in the lower fields, reference to which has already been made.

Leading Anthracite Breakers

The preparator, commonly known as the breaker, will be discussed, to show how the features mentioned are assembled to obtain the best results. The breakers described show the practice in practically the entire anthracite field. Two of these breakers are in the Scranton region, one is in the Wilkes-Barre, one in Nanticoke, one in Hazelton, one near Mahanoy City, one near Lykens, and one in the Panther Creek Valley.

Marvine Breaker, Hudson Coal Co.

In 1920, construction was started on a 5000-ton steel breaker at the Marvine colliery, of the Hudson Coal Co., in order to concentrate in one breaker the preparation of material that was being handled in two old structures wherein the dry method of preparation was in use. Besides, the old Marvine breaker was unable to handle the tonnage that the mines could produce.

The Manville breaker, one of the two eliminated by this concentration, is situated about 1 mi. from the Marvine. The coal is now dumped in this old plant and run through a pair of rolls, which crushes it to steamboat size, then by chutes it is delivered into railroad cars that convey the coal to the new Marvine breaker, where it is dumped into a conveyor line.

The Marvine has two hoisting shafts 2000 ft. (609.6 m.) apart, but one of these was used only to hoist the coal from the lower to an intermediate level, where it was sent to the main shaft up which it was hoisted into the breaker. As the new breaker can handle the output from both shafts, the output is practically doubled.

One interesting feature of this new breaker is that the coal from one of the shafts is carried to the breaker over the main line of the Delaware & Hudson R. R. and across the Lackawanna river; two belt-conveyor lines, approximately 1100 ft. (335 m.) in length, Fig. 39, transport it in this latter portion of the journey.

The new Marvine breaker is constructed of steel and prepares the coal by the wet method. The building is as nearly fireproof as it can be made. The only wooden construction is the jigs, the inside lining of the loading pockets, the treads of the stairs, the shaker sides, hangers, and arms, the

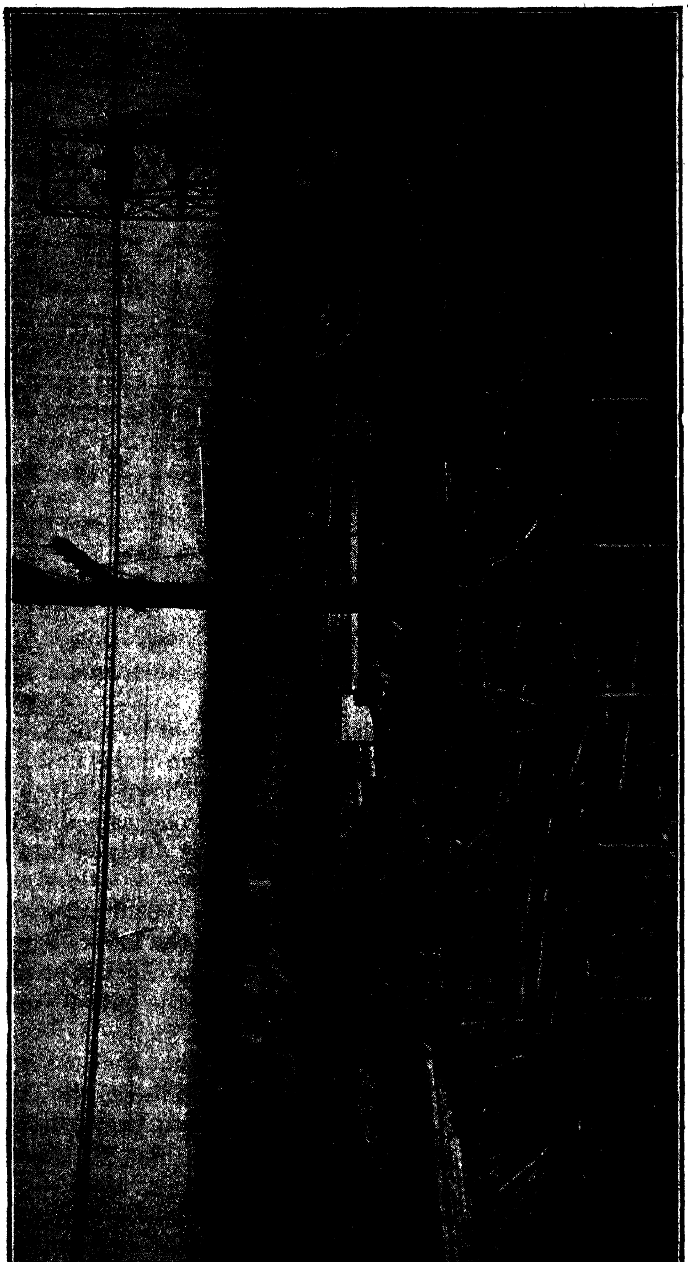

Fig. 39.—View of long conveyor lines that transport coal from head houses to Marvine breaker; these conveyor lines are 1100 ft. long.

slate-conveyor trough, and the troughs on the three main conveyor lines. The breaker is electrically operated throughout and controlled from a central switchboard. It is equipped with 44 Delaware, or Tench, piston-type jigs, and a complete plant for the treatment of the silt is installed nearby. The latter consists of Dorr thickeners and classifiers and Deister-Overstrom concentrating tables.

The coal is crushed on the ground level before it is taken into the breaker, so that the only crushing done is that of the grate, or broken, coal when no market can be found for this size. Crushing the coal on

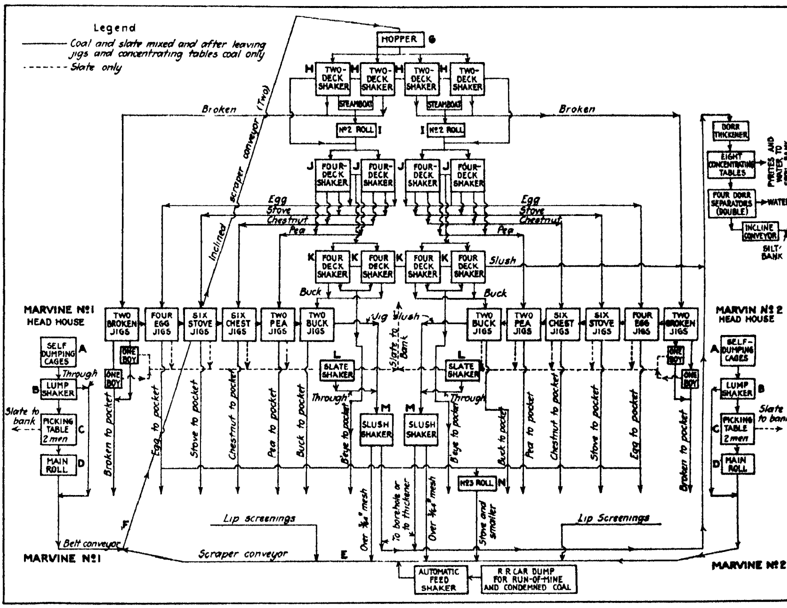

FIG. 40.—FLOW SHEET OF MARVINE BREAKER.

the ground level has the advantage of eliminating the heavy crushers and bull shakers at the top of the building, which cause severe stress on the structure, and permits a considerable reduction in the height of the structure. Another interesting detail is the complete elimination of coal-carrying elevators. Water is supplied to this breaker from the Lackawanna River by electrically driven pumps.

This breaker is constructed in two distinct units; that is, it is so built that either half of the breaker is a complete operating unit and can be shut down without interference with the running of the coal through the other half.

The following is a description of the flow of coal through the breaker and the method of preparation followed, Fig. 40.

The two head houses *A*, situated at the top of the two hoisting shafts, are identical in construction. Coal is hoisted from each shaft, which contains two hoisting compartments in which self-dumping cages operate. The coal is dumped into a chute, which delivers it to the lump shaker *B*. The lump-size coal passes from this shaker on to a gravity picking table *C*, where two men remove the rock, which is sent to the slate bank. The coal passes through the main rolls *D*, which crush it to steamboat size and smaller. The material passing through the lump shaker *B* is conveyed by chutes to a point under the rolls *D*, where it mixes with the material from the rolls.

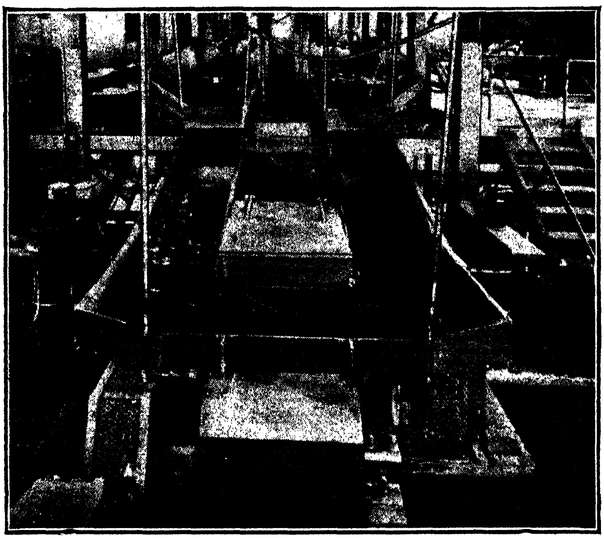

FIG. 41.—GRATE AND EGG SHAKERS IN MARVINE BREAKER.

From head house No. 1, the coal is transported by means of the two belt-conveyor lines for a distance of approximately 1100 ft. to the inclined scraper-conveyor lines *F*. The coal from head house No. 2 is moved by a scraper conveyor *E*, which travels directly underneath the center of the breaker. Into this is delivered, as it passes under the building, all material such as products of the rolls breaking egg coal, material from the slate shaker, that from the slush shaker, and from the lip screens. This conveyor also receives the material dumped from railroad cars, either run-of-mine, previously crushed to steamboat size, and condemned coal, both of which are fed to this conveyor by an automatic feed. This conveyor line delivers its material to the inclined scraper conveyors *F*, each of which is designed to handle the entire tonnage of this breaker. These conveyors deliver the material to a hopper *G* at the top of the

building, thence the material passes to four double-deck shakers *H*, Fig. 41. The steamboat material passes from the deck of these shakers into the No. 2 rolls *I*, where it is crushed to egg and smaller. The material passing from the second deck of the shakers *H*, which is the broken, or grate, size, is sent either to the No. 2 rolls *I*, where it is crushed to egg and smaller, or, when a market exists for this size, it goes to two jigs on each side of the breaker from which the coal product, after passing a picker boy, goes to the loading pocket, and the slate product, after passing a picker boy, goes to the slate bank. Experience has shown that it is necessary to employ one boy on the slate and one on the coal discharged from each of the jigs in order properly to prepare this coal for the market and maintain the slate free from coal.

Material passing through the shakers *H*, being egg coal and smaller sizes, is mixed by chutes with the product of the No. 2 rolls *I*. This material then passes on to four sets of four-deck shakers *J*, which size the coal into egg, stove, nut, and pea. The egg coal, which comes from the top deck, goes to four jigs on each side of the breaker. Washed coal from these jigs goes directly to the loading pocket and the slate to the slate bank, both without any hand picking. In case egg coal is not in demand, this size after leaving the jigs may be passed to the egg-coal rolls *N* which break it down to stove and smaller sizes; the material from these rolls passes into the main intake conveyor underneath the breaker.

Stove coal, coming from the second deck of these shakers, goes to six jigs on each side of the breaker. The washed coal from each jig passes to the loading pocket and the slate to the slate bank, both without picking. Chestnut coal, from the third deck, goes to six jigs on each side of the breaker; as in the case of the other sizes, the washed coal goes to the loading pocket and the slate to the slate bank. Pea coal, from the fourth deck, goes to two jigs on each side of the breaker and, as before, the coal product of these machines goes directly to the loading pocket and the slate is sent to the slate bank.

Material passing through these shakers *J*, consisting of No. 1 buckwheat and smaller sizes, goes to the 4 four-deck shakers *K*, which make No. 1, No. 2, No. 3, and No. 4 buckwheat, the last three sizes being mixed and shipped as bird's-eye. No. 1 buckwheat comes from the upper deck and passes to two jigs on each side of the breaker; the washed coal from these machines goes to the loading pocket and the slate to the slate bank. No. 2 buckwheat, from off the second deck, No. 3 buckwheat, from the third deck, and No. 4 buckwheat, from the fourth deck, mix at the end of the shakers and the resulting bird's-eye is conducted, by chutes, to the loading pocket. The slush, or material which passes through all decks, is conducted to a separate building for further treatment.

All slate from the jigs pass over slate shakers *L* to reclaim the fine

breakage. The material going over these shakers passes to the slate bank; that passing through them joins with the slushings from the jigs. This mixture then passes over the slush shakers *M*. The material passing over a 3/64-in. (1.2 mm.) mesh goes into the main conveyor line *E* underneath the breaker. The material going through these slush shakers passes to the plant for the treatment of the slush.

Lip screenings from all the loading pockets, Fig. 42, go to the main conveyor line *L* under the breaker. The slush-treatment plant, which

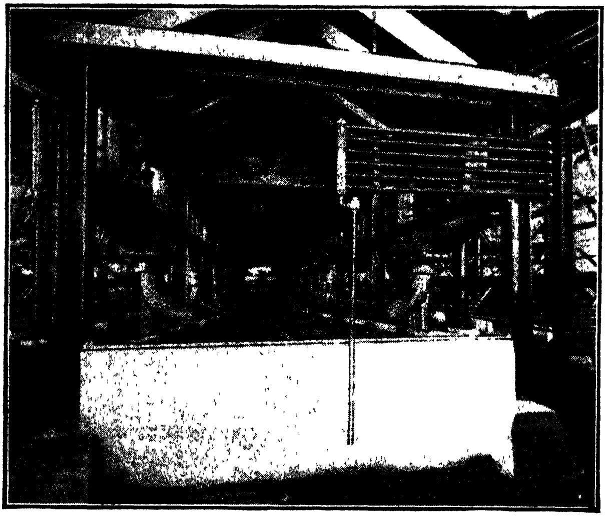

FIG. 42.—BOTTOM OF LOADING POCKETS OF MARVINE BREAKER.

receives all the slush from the breaker, consists of a Dorr thickener, in which the slush is settled out of the water; that which overflows contains only the smallest particles of the suspended solids. The thickened material from these machines is fed to eight concentrating tables and the coal from these passes to four Dorr separators where a large percentage of the water is removed. The coal is then conveyed to a stock pile or a loading pocket for shipment.

Pyrite from the concentrating tables may be recovered or discarded as desired. The water from the Dorr thickener and separator passes out of the plant.

No. 1 *Breaker, Pennsylvania Coal Co.*

At No. 1 colliery of the Pennsylvania Coal Co. at Dunmore, just outside of the city of Scranton, a new breaker is being constructed. This also is a steel and concrete structure; several details, however, vary

greatly from those in the Marvine breaker. Fig. 43 is a flow sheet of this breaker. When the coal leaves the conveyor, it will be dumped into a chute leading to the three-deck main shakers 4 in the top of the breaker. The lump coal, which includes the steamboat, will pass to a picking chute 5 and the grate, broken, and egg will go Elmore jigs 6 and 7. The rock from these jigs is to be hand-picked to remove the coal and bone, the

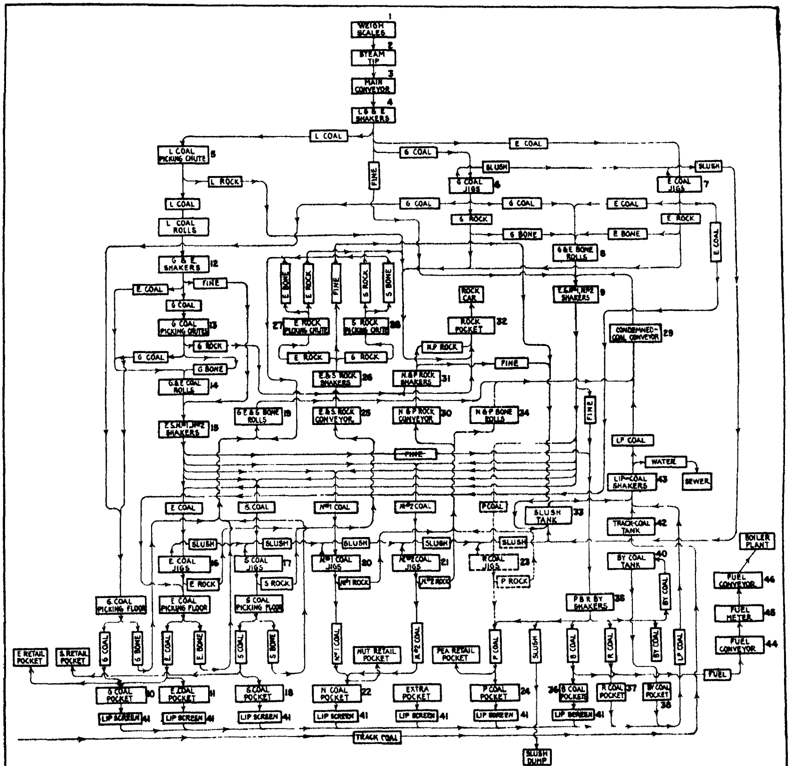

FIG. 43.—FLOW SHEET OF NO. 1 BREAKER OF PENNSYLVANIA COAL CO.

latter being sent to the bone rolls 8, from which the material will pass to a shaker 9, which makes egg, stove, and two sizes of chestnut coal. The coal from the jigs will then pass to the picking floor, where the bone left in the coal will be removed. The cleaned product goes to the pockets 10 and 11.

Cleaned lump coal from the picking chute 5 will go through the rolls, thence to a set of broken or grate-and-egg shakers 12. Grate coal will pass to a picking chute 13 and then unite with the egg coal and pass through rolls 14. Thence it will go to another set of shakers 15 on

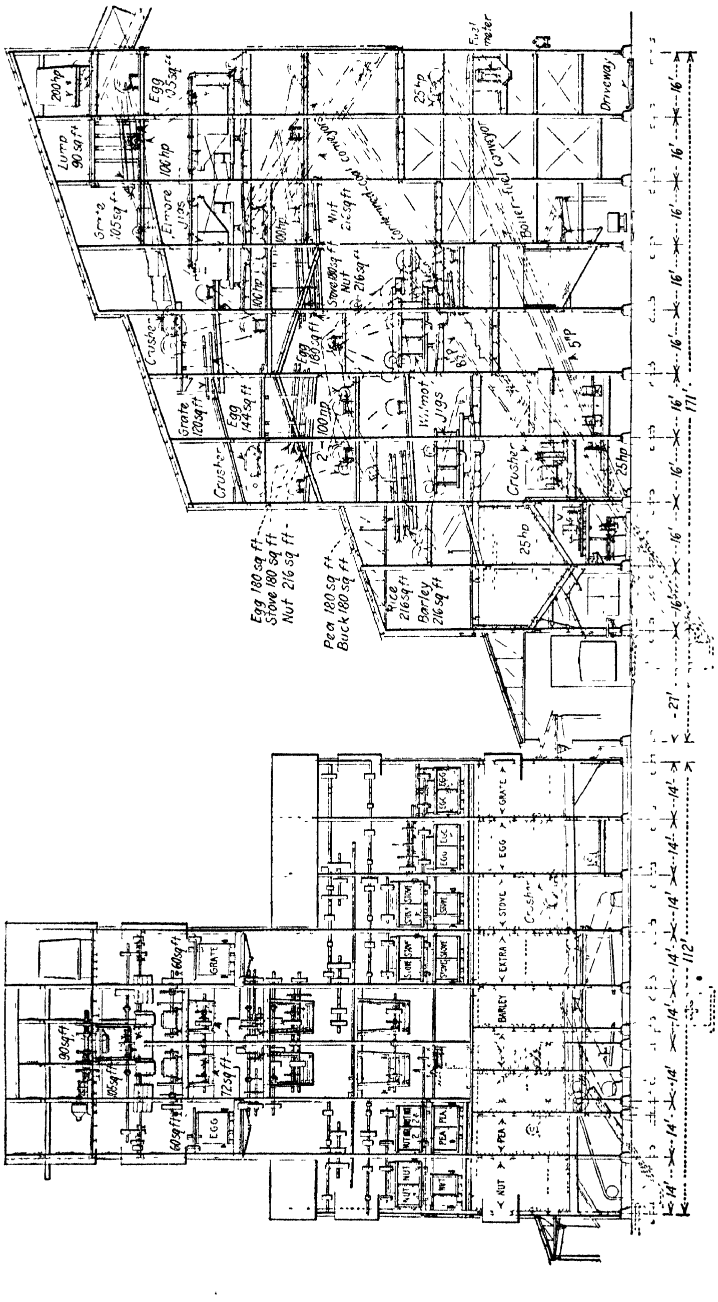

FIG. 44.—FRONT AND SIDE ELEVATIONS OF NEW NO. 1 BREAKER OF PENNSYLVANIA COAL CO. THIS SHOWS AREAS OF SCREENS AND HORSEPOWER NECESSARY TO OPERATE BREAKER.

which egg, stove, and two sizes of chestnut coal are made. The egg, stove, and nut from shakers will go to Wilmot jigs. After cleaning, coal from the egg jigs 16 and the stove jigs 17 is to be hand-picked, the product going to the proper pockets 11 and 18. Bone coal recovered in hand-picking the grate, egg, and stove coals will unite and go through rolls 19, the crushed product of which will be carried by the condemned-coal conveyor 29 to shaker 9.

Cleaned coals from jigs 20 and 21, which will treat the two nut sizes, unite and go to the nut pocket 22. Shaker 9 is so arranged that the bottom deck can be changed to produce pea coal; in that case this size will pass to the pea jigs 23 the cleaned product of which will go to the pocket 24.

Rock from the egg and stove jigs 16 and 17 unites and is to be taken by the egg-and-stove rock conveyor 25 to the egg-and-stove rock shakers 26. Here it is to be separated, after which the bone will be hand-picked (27 and 28) from the rock and sent to the grate-egg-and-stove bone rolls 19, thence to the condemned-coal conveyor 29. The rock from the nut and pea jigs 20, 21, and 23, will unite and go to the nut-and-pea rock conveyor 30, thence to a shaker 31 where the fines are to be removed. The rock will go to the rock pocket 32 and the fine to the slush tank 33. Instead, however, of sending this rock from the nut and pea jigs to the rock conveyor 30 it can be sent to the nut-and-pea rolls 34 from which the resulting product is sent to the condemned-coal conveyor 29.

All the fine coal from shakers 9 and 15 unites and passes to shaker 35; here pea, No. 1 buckwheat, rice, barley, and slush are separated. The sized coals will not be further treated but will pass directly to their respective pockets 37 and 38, the slush going to the slush dump 39. The barley coal can be sent from the shaker 35 to the barley tank 40 instead of to the pocket. From this tank it passes to the pocket 38. All the slush from the Elmore jigs near the top of the breaker and from the Wilmot jigs will pass to the slush tank 33.

In loading railroad cars, an appreciable amount of coal is spilled around the tracks. At many breakers, this is lost. At this breaker a concrete floor has been placed all around the car tracks and suitable drains made so that it will be possible to flush this whole area with water, thus washing the coal into drains that will conduct it to the track-coal tank 42. All the lip-screen coal 41 will be taken to a shaker 43 on which it is to be joined by the track-tank coal. Here the water will be separated from it. The coal is then to be delivered to the condemned-coal conveyor by which it is to be taken back into the breaker for retreatment.

The buckwheat, rice, and barley sizes, instead of going to their respective pockets after being screened on shaker 35, can be sent to a fuel conveyor 44, which will deliver its material to the fuel meter 45, after which it is taken by another conveyor 46 to the boiler plant.

Fig. 44 shows the position of the machinery, also the screen area and the horse power of the motors necessary to drive the various pieces of equipment. This breaker has, what is to all purposes, an individual electric drive. One motor seldom actuates more than one, or at the most two, pieces of machinery at a time. A total of 875 hp. is required to drive all the machinery.

Franklin Breaker, Lehigh Valley Coal Co.

The two breakers just described use the wet type of treatment; the Franklin breaker of the Lehigh Valley Coal Co., located near Wilkes-Barre, Pa., employs both wet and dry methods. Its drive system also is different, a single power unit being used to actuate the machinery.

Fig. 45.—Exterior of Franklin breaker of Lehigh Valley Coal Co. at Wilkes-Barre, Pa.; steel frame and concrete floors, together with sheet-iron siding, make breaker practically fireproof.

Another marked change is that the coal pockets, instead of being placed above the loading tracks, the coal being drawn off by gravity, are placed at right angles to the tracks and level with or below them. A belt conveyor between the two rows of pockets transports the coal to the railroad cars.

With this arrangement the height of the breaker is reduced, less labor is required in loading the cars, the cost of the construction of the breaker is lessened, and its foundations need not be so heavy, for the

weight of the pockets and of the coal in them is removed from the breaker foundation.

The general layout of the pockets at a simiarly equipped breaker is shown in Fig. 53. All rock separated in the Franklin breaker is pulverized and returned to the mine for silting purposes.

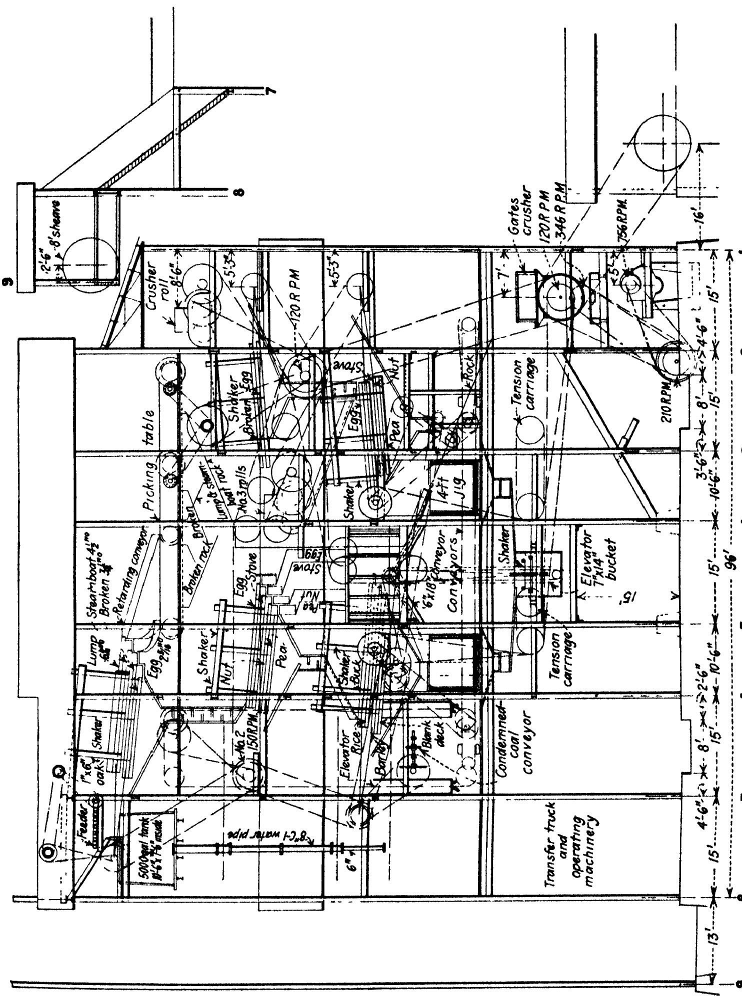

FIG. 46.—ARRANGEMENT OF MACHINERY IN FRANKLIN BREAKER OF LEHIGH VALLEY COAL CO.; LOADING POCKETS ARE SIMILAR TO THOSE SHOWN IN FIG. 53.

The breaker is built of steel and covered with sheet iron; it is well provided with windows, as shown in Fig. 45; Fig. 46 is an elevation and Fig. 47 a flow sheet of the breaker.

The mine cars are hoisted to the top of the breaker and the coal dumped into a hopper, from which it is fed by a reciprocating feeder on to the four-deck shakers. Lump coal, taken off the top deck, is hand-picked to remove the rock. Steamboat coal, taken off the second deck, is likewise hand-picked, the cleaned coal uniting with the cleaned lump. Both sizes then pass through a set of No. 1 rolls on to a set of four-deck shakers. From the top deck, broken, or grate, coal is taken. This size can be sent either to its pocket or to a set of breaking rolls, from which the coal passes to another shaker. From the top deck of this, egg is

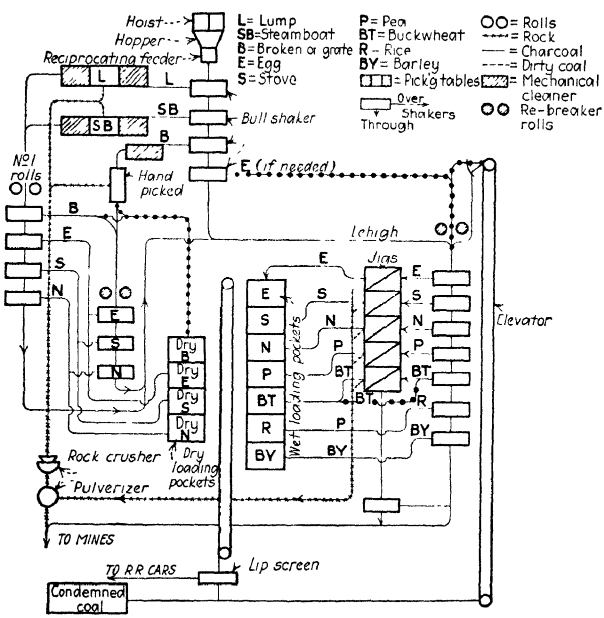

FIG. 47.—FLOW SHEET OF FRANKLIN BREAKER.

taken; this unites with the egg from the second deck of the shaker fec from the No. 1 rolls and passes to the egg pocket. Stove coal, taken fron the second deck of the broken-coal shaker, unites with the same siz from the third deck of the shaker fed from the No. 1 rolls. Nut of tw sizes from the third and fourth decks of the two shakers, respectively unite in the same way and pass to the appropriate pocket. This con stitutes the dry-coal side of the breaker, no water being used up to thi point. From the third deck of the bull shaker, broken coal is taken this goes through a set of mechanical pickers and is also hand-picked t remove the slate. The cleaned broken coal goes either to its pocket o through the re-breaking rolls.

The dirty coal passes through the bottom deck of the bull shaker and is wet. If it is desired to break down the egg coal, another deck is employed on the bull shaker, the egg coal being separated at this point and sent to re-breaking rolls. If on the other hand it is not desired to re-crush this size, this deck is not used and the coal from the bull shaker unites with that from the shaker that is fed from the No. 1 rolls and with that from the shaker that is fed from the broken re-breaking rolls. The coal from these three sources passes to a set of fine-coal screens, on which egg, stove, chestnut, pea, No. 1 buckwheat, rice, and barley are made. This coal is sent to reserve pockets, from which it is fed into the jigs.

Lehigh Valley jigs are used to prepared egg, stove, chestnut, pea, and No. 1 buckwheat. The rice and the barley sizes go to their respective pockets without jigging. Slush or culm is sent to the mines for silting purposes. The rock from the two picking tables at the top of the breaker is sent to a Gates rock crusher, after which it unites with the rock from the jigs and is pulverized so that it may be sent into the mine as flushing material.

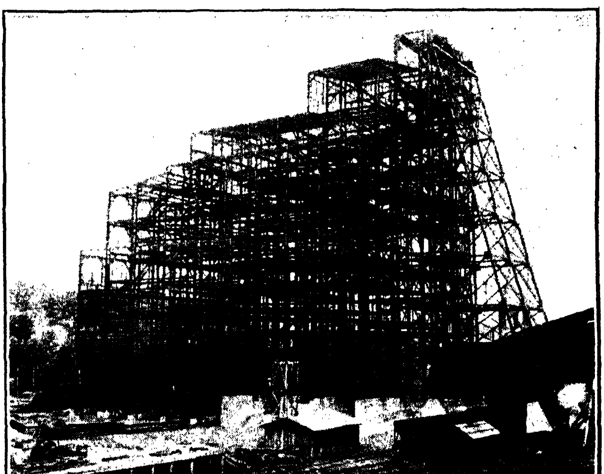

FIG. 48.—WANAMIE BREAKER OF LEHIGH & WILKES-BARRE COAL CO. UNDER CONSTRUCTION.

Wanamie Breaker, Lehigh and Wilkes-Barre Coal Co.

In the Wanamie breaker, Fig. 48, may be found methods not adopted in any of the breakers already described. All sizes above egg are prepared dry; egg and smaller sizes are prepared wet. The first difference noticeable in this breaker is the grizzly used to remove the lump coal from the run-of-mine. The second variation is the use of anthracite

spiral pickers for the primary removal of slate from the stove and egg coals. The building is of steel construction and is as near fireproof as possible.

On the flow sheet, Fig. 49, are presented the results of a run of 200 mine cars in 1 hr. and the amount of coal that passed through each process. Slate was not taken into account. As the breaker is double, the

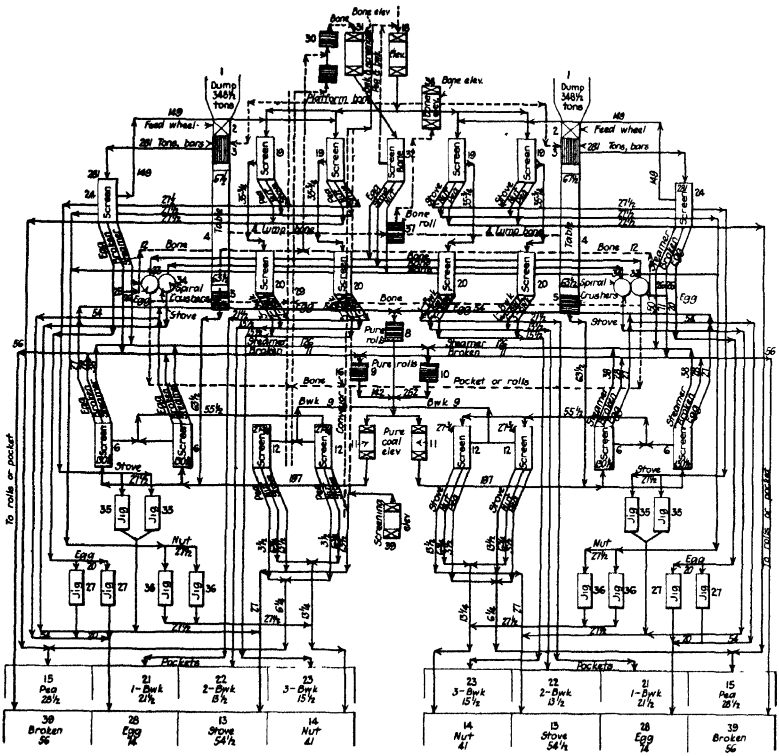

Fig. 49.—Flow sheet of Wanamie breaker; the two halves of this breaker are exact duplicates.

method of preparation of the coal is duplicated on the two sides, but the cleaning of the bone is not a double operation. The flow sheet may be easily compared with the elevation of the breaker shown in Fig. 50.

Coal is discharged into chutes from the mine cars, which are hoisted to the top of the building; thence it is fed by a rotary feeder 2 on to grizzlies 3. Sizes smaller than lump pass through the bars, the lump coal passing over on to picking tables 4, where the rock is removed. In the test shown, 67½ tons of material passed over the table and 63½ tons of

coal were sent to the rolls, showing that 4 tons of worthless material was removed. This was duplicated, of course, on the other table.

After passing the rolls 5, the crushed coal goes to two sets of shaker screens on each side of the breaker, where the egg, broken, and steamboat sizes are removed. The steamboat and the broken, as well as the egg coal if it is so desired, go through the re-breaking rolls (8, 9, and 10) and then are returned by the elevator 11 to the same set of screens for resizing.

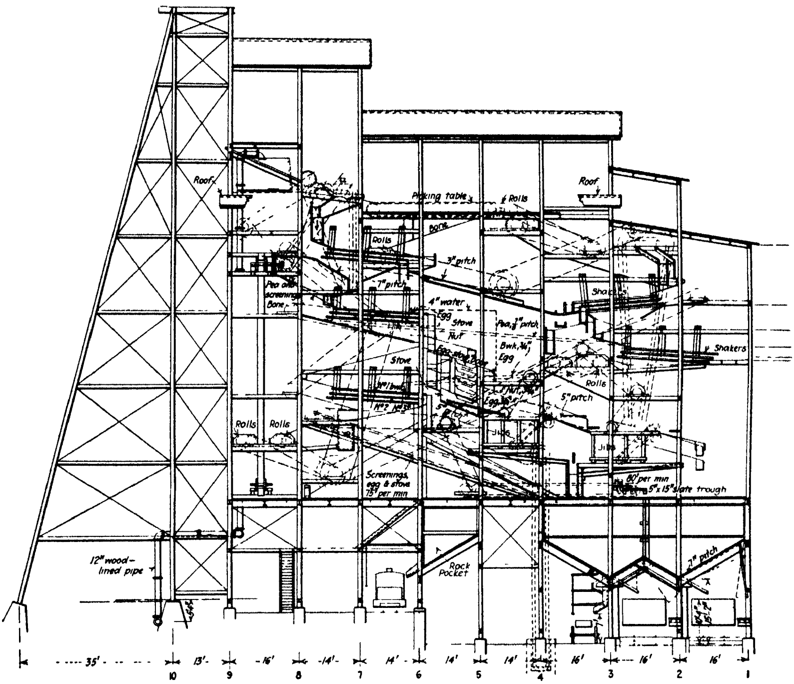

FIG. 50.—ELEVATION OF WANAMIE BREAKER SHOWING ARRANGEMENT OF MACHINERY.

Egg and smaller, if it is not broken down, passes to another set of screens where stove, chestnut and pea are made, these sizes going directly to their respective pockets 13, 14, and 15. The finer sizes, which go through the lower deck of these last shakers, are taken by a conveyor 16 to an elevator 18 and raised to another set of shakers 19, on which any remaining pea, stove, or chestnut sizes are removed. The fine coal goes to another set of shakers 20, on which No. 1 buckwheat, No. 2, or rice, and No. 3, or barley, are made. These sizes then go to their proper pockets, 21, 22, 23, without further treatment.

The fine coal that passed through the bar screens at the head of the breaker goes to sets of shakers 24 where steamboat, broken, and egg

coals are separated. The first two of these sizes are hand-picked (25, 26), the cleaned product going to the re-breaking rolls (9 and 10), already mentioned. The process followed in the subsequent treatment of these sizes is the same as has been described. Egg coal, from the set of shakers last mentioned, passes to the egg jigs 27 and the cleaned product goes to the egg pocket 28.

Bony coal separated on the broken and steamboat picking tables 25 and 26, is sent, by conveyor 29, to the bone rolls 30, from which it goes to the bone elevator 31, which delivers it to the bone shaker 32, where egg, stove, and chestnut grades are separated. The egg and stove coals from this shaker pass to anthracite spiral pickers 33, 34, whence the clean egg coal goes to the egg pocket 28 while the bony coal passes to the bony conveyor 29 and is delivered to the bony rolls 30 for recrushing and subsequent re-treatment. The same process is followed in the treatment of the coal and bony from the stove spirals 34. Stove, chestnut, and pea coals made on the shakers 19 pass to the stove and nut jigs 35, 36, the pea size going to its pocket 15. Cleaned sizes from the jigs likewise pass to their respective pockets 13, 14. Fine coal passing through these shakers goes on to another set of screens 20 and is separated into No. 1 buckwheat, rice, and barley. These are not further treated but go to their respective pockets 21, 22, and 23.

Bone coal removed by hand from the picking table 4 passes to a bone roll 37, thence to a bone elevator 38 by which it is discharged on to the bars 3 at head of breaker. The condemned-coal elevator 39 takes the coal to conveyor 16, which discharges it into an elevator 18 which takes it to shaker for retreatment. In case there is a demand for broken coal, instead of sending this size, as made on shaker 6, to the rolls to be rebroken, it can be sent direct to the broken pocket 39.

Lattimer Breaker, Pardee Brothers & Co., Inc.

The conditions under which coal is produced to a large extent determine the methods that must be followed in its preparation. In the Lattimer breaker, of Pardee Brothers & Co., Inc., near Hazelton, a condition is encountered that is somewhat different from that ordinarily found. Large strippings are located at this operation so that the coal is apt to be of large size, as it is loaded into the mine cars by steam shovels.

Fig. 51 is the flow sheet of this breaker. The coal is dumped into a pocket at the top of the building and is passed by an automatic feeder to the bull shakers. The large lump coal going over the top of this shaker passes to a picking table, where both the rock and the large lumps coming from the strippings are removed. Steamboat coal from the bull shaker passes to a picking table, after which the clean steamboat unites with the clean large lumps from the first table and passes through a set of rolls. It then runs on to a set of shaker screens on which broken, egg, stove,

chestnut, pea, and No. 1 buckwheat grades are separated. As this coal s already clean, it passes directly to the proper pockets. Should it be desired to re-break the broken coal, it can be elevated and sent back through the rolls to be crushed to finer sizes instead of being chuted directly to its pocket.

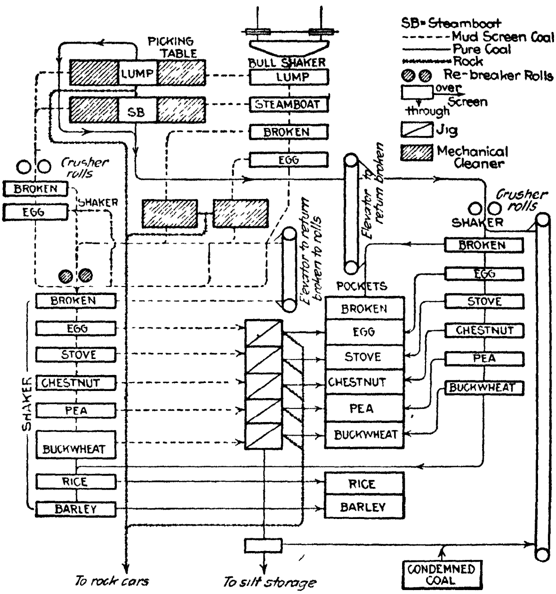

FIG. 51.—FLOW SHEET OF LATTIMER BREAKER OF PARDEE BROS. & CO., INC.

Coal passing over the picking tables in the head of the breaker is united and goes to a set of rolls, thence on to a set of shakers from which the broken and egg coals go to a set of re-breaking rolls to be broken to finer sizes. Should it not be desired to break the egg coal to smaller sizes, this grade is bypassed around the rolls to another set of shakers. The coal that comes through the rolls likewise passes to the same set of shakers, where it is joined by other coal.

On the third deck of the bull shaker, broken coal is removed and on fourth deck egg coal is taken off. These coals both pass to a set of mechanical pickers where some of the rock is removed. The coal from these pickers unites with that mentioned above, as does the coal of similar size that passes through the bull shakers. All the coal is thus re-united, except the clean material removed from the original picking tables. This coal is now sized. The broken coal taken off the top deck of the shaker is elevated to the rolls to be re-crushed. The egg, stove, chestnut, pea, and No. 1 buckwheat pass to the jigs for preparation, thence to their respective pockets.

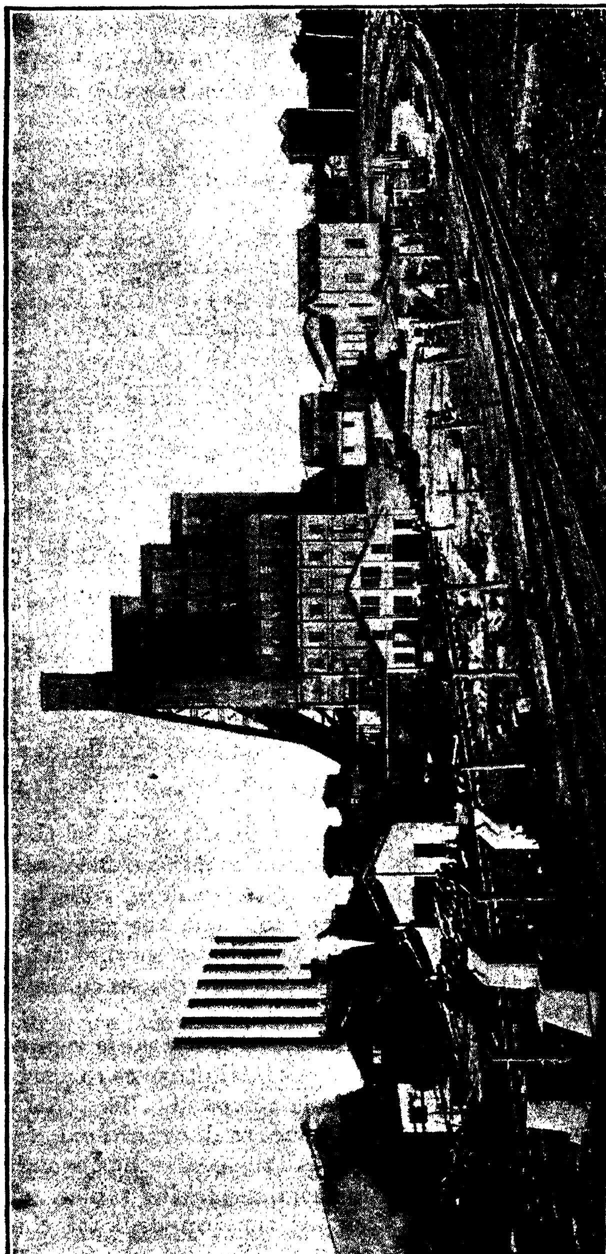

FIG. 52.—VIEW OF BUCK MOUNTAIN COLLIERY OF LEHIGH VALLEY COAL CO., ALSO SHOWING BREAKER.

Coal finer than No. 1 buckwheat, passing through the bottom deck of the clean-coal shakers, unites with the coal of similar size from the mud screen and passes over a shaker, which separates the rice from the barley. These grades are then ready to pass to their respective pockets.

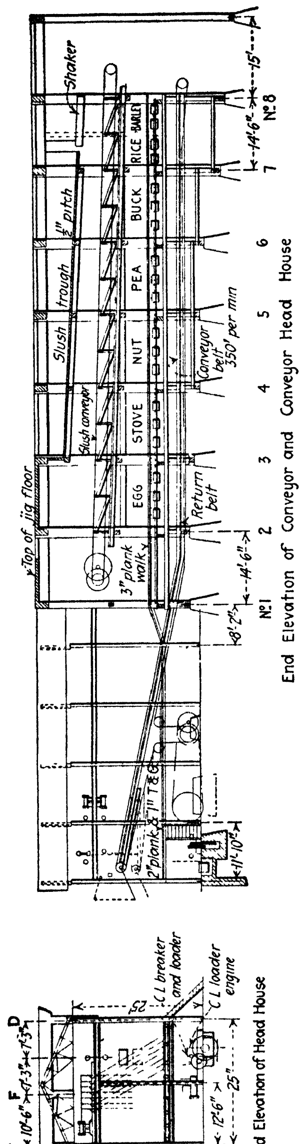

FIG. 53.—ARRANGEMENTS FOR LOADING COAL AT BUCK MOUNTAIN BREAKER.

All rock from the picking tables, mechanical separators, and jigs goes to the rock pocket, whence it is transported by rock cars to the dump. The silt from the jigs and from the bottom deck of the shakers goes to the silt storage pile.

Buck Mountain Breaker, Lehigh Valley Coal Co.

The construction of the Buck Mountain breaker, Fig. 52, is similar to that of the Franklin, owned by the same company. The same arrangement of loading pockets is used; the layout is shown in Fig. 53. A belt conveyor carries the coal from the pockets to a chute at a sufficient height above the tops of the railroad cars that the coal will slide into the cars by gravity.

This breaker lies in the southern middle field where the mining conditions differ from any hereinbefore discribed; as a result, the method of preparation is different also. The coal in this field lies in steep-pitching measures that are quite wet; in fact when a car of coal comes from the mine it looks more like a car of mud than one of coal.

The flow sheet, Fig. 54, shows that the coal is hoisted to the top of the breaker, in the mine cars, and is then dumped into a hopper, from which it is fed on to the bull shakers by a manually operated gate. On the upper deck of this screen, the lump coal is separated passing thence on to a picking table where the rock is removed. From this table, the lump coal passes to a set of crushing rolls, where it is

broken down. The resulting cleaned coal is then passed over a shaker, the broken coal going to a set of re-break rolls, thence into a hopper. Egg coal from the same shaker either goes directly to the same hopper or passes to the rolls.

Broken coal from the second deck of the bull shaker is crushed to egg and smaller and then passes to the hopper above mentioned. Egg coal from the third deck of the bull shaker goes either to the broken

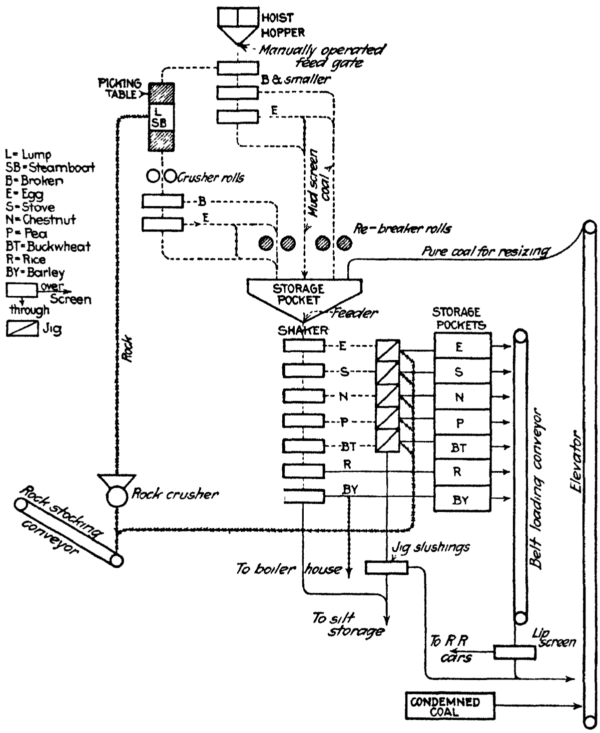

FIG. 54.—FLOW SHEET OF BUCK MOUNTAIN BREAKER.

rolls or directly to this same hopper. Thus all of the coal is re-united and broken down to egg size after the large rock has been removed from it. From the hopper, the coal is delivered by a feeder to a set of shaker screens by which it is sized into egg, stove, chestnut, pea, No. 1 buckwheat, rice, and barley. The first five sizes are sent to jigs and cleaned of their rock, the cleaned coal going to its proper pocket. The last two sizes are sent to their pockets without further treatment, except that the barley may be sent to the boiler house to be used as fuel. The jig slushings and the material passing through the lip screen now

go to another shaker, from which the large-size material goes to the condemned-coal elevator, while the fine slush goes to the silt storage.

All the large rock is sent to a Gates crusher, by which it is broken down in size. It is then joined by that from the jigs and taken by a rock-stacking conveyor to a waste bank.

Brookside Breaker, Philadelphia & Reading Coal and Iron Co.

The Brookside breaker of the Philadelphia & Reading Coal and Iron Co., near Tower City, in the southern coal field, embodies features different from any hitherto described. In the first place, it is constructed of wood. As shown in Fig. 55, it is what might be called a hillside breaker; that is, in its construction, advantage has been taken of the natural contour of the ground. This permits a great saving in material. Fig. 56 is the flow sheet.

The preparation of the fine sizes is more careful than in most of the breakers hitherto described. In the mining of coal in steep-pitching measures, practically all the material produced in the mine, whether rock or coal, is brought to the surface. In this case, the mine cars, regardless of the material with which they are loaded, are brought to the breaker head. Two dumps are provided, one for rock and one for coal cars. The rock cars are discharged into the rock chute, which delivers to a rock shaker 1. The coal cars are discharged into a chute 2, from which the coal goes to the bull shaker 3. Here the lump is taken off the top deck and goes to a picking platform 4, which is of the gravity type, where the pure lump coal is separated from the bone and rock. Rock from this platform unites with that from the rock-picking platform under the rock shaker 1 and the coal recovered from that platform unites with the bone from the picking platform 4.

The pure lump coal from picking table 4 passes to a set of No. 1 rolls 5 and is crushed to steamboat and smaller. It then goes to a shaker 6, where the steamboat is taken off the deck, the finer material passing through. The pure steamboat then goes to a set of No. 3½ rolls 7 and is crushed to broken and smaller. The coal from these rolls unites with that passing through the steamboat shaker 6 and goes on to the broken shaker 8; here the broken coal is separated, going to the broken pocket. If, however, sufficient demand for this coal does not exist, it passes through a set of No. 3 rolls 9 and then unites with the coal from shaker 8 and passes to shaker 10, where the egg is removed and sent to its pocket; if it is desired to break this coal still further it passes through a set of No. 4 rolls 11 and the coal then unites with the fine coal from shaker 10 and passes to shaker 12, on which stove, chestnut, pea, and No. 1 buckwheat are made, each size going to its proper pocket.

Coal that passes through the lump-coal shaker 3, goes to the main hopper 13 thence to a shaker 14. From the top deck of this shaker, the

steamboat coal is taken, passing to a picking table 15 from which the pure coal goes to a set of rolls 7. The rock from the picking table 15 goes to the rock pocket. From the lower deck of the same shaker, broken coal is taken. This goes to a picking chute 16, from which the pure coal

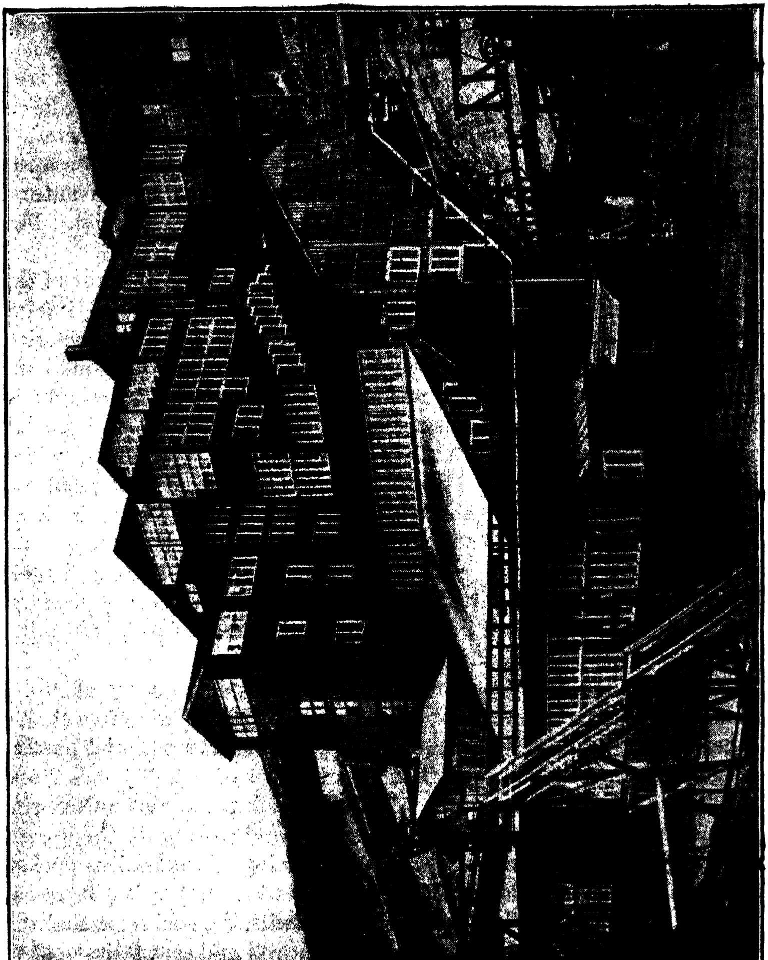

Fig. 55.—Brookside breaker of Philadelphia & Reading Coal & Iron Co.

either goes to the pocket or to the rolls 9. The bone coal from both the steamboat and the broken picking chutes goes to a set of No. 3 rolls 17, and thence to the main elevator.

Fine coal passing through shaker 14 goes to another set of shakers 18, from the top deck of which egg coal is taken. This passes to an egg

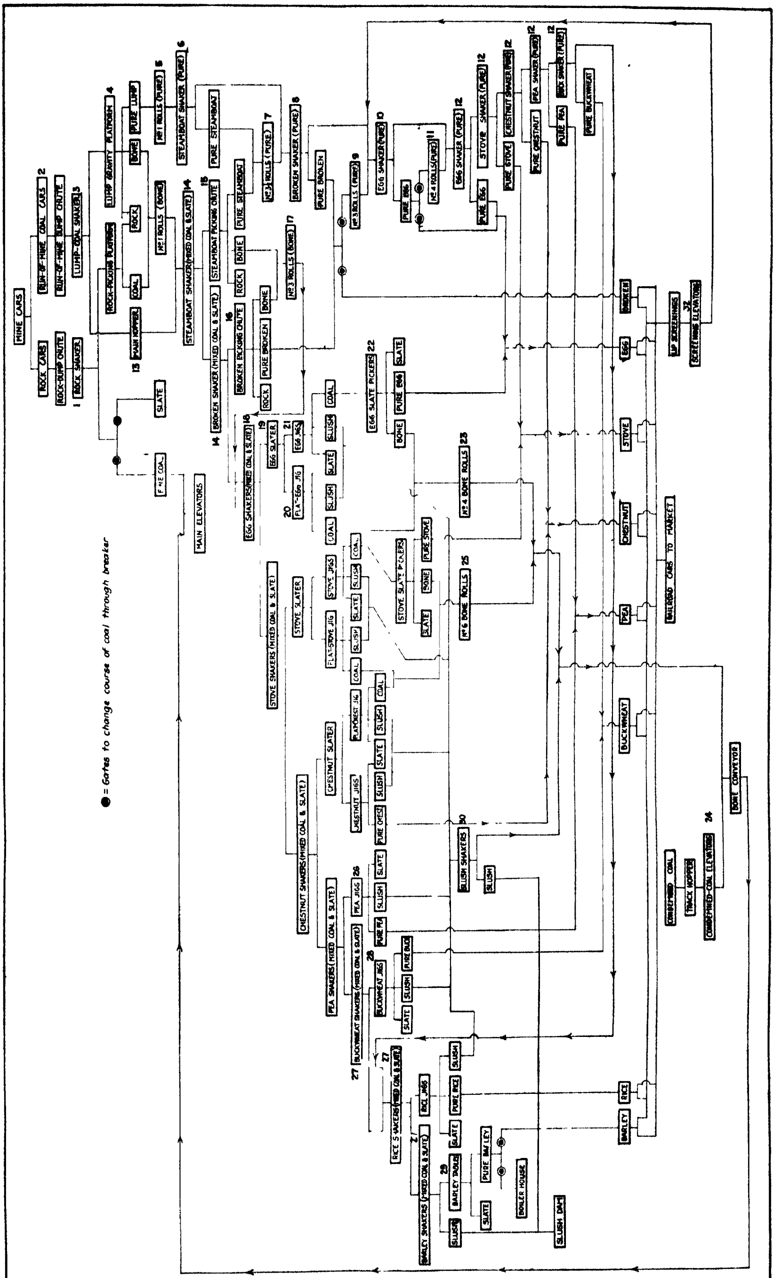

FIG. 56.—FLOW SHEET OF BROOKSIDE BREAKER.

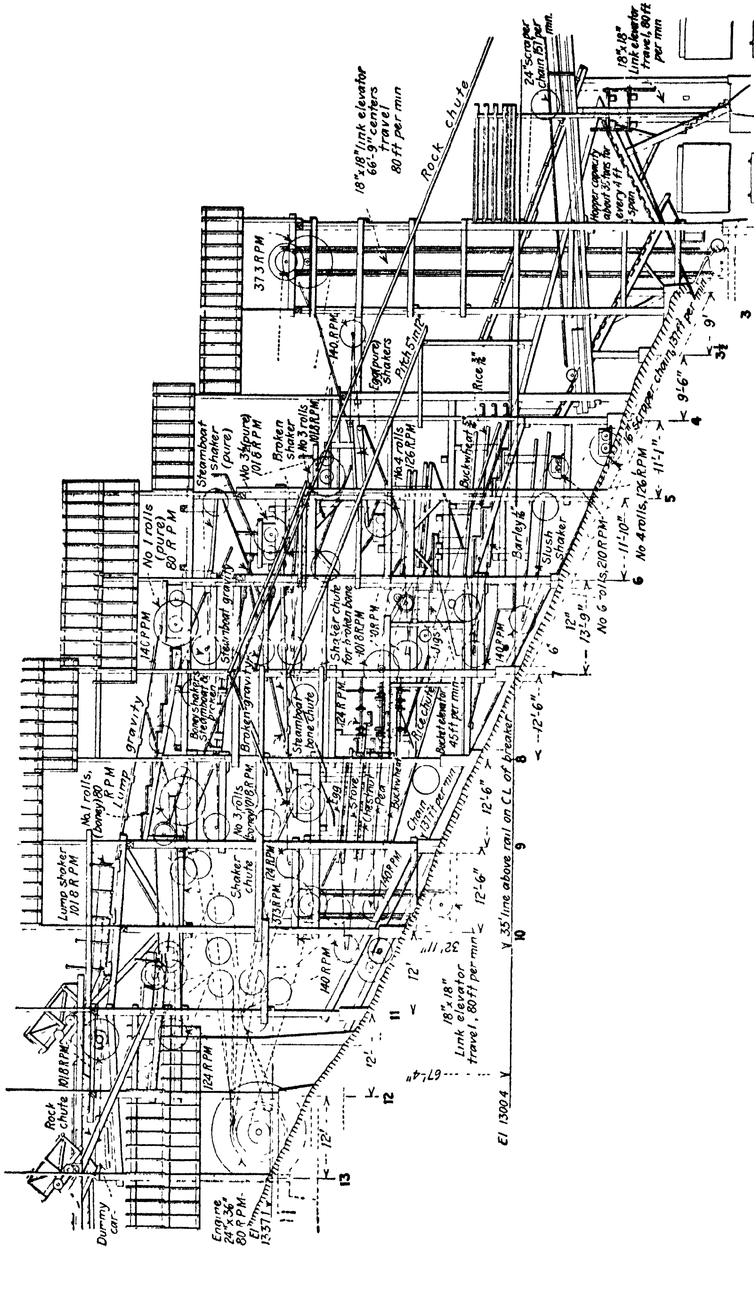

FIG. 57.—SIDE ELEVATION OF THE BROOKSIDE BREAKER.

slater 19, which separates this size into flat and round pieces. These then go to separate jigs 20, 21. The cleaned coal is picked at 22 to remove any slate that may remain, and the resulting product goes to the egg pocket. Bone from jig 20 and that which is picked after treatment from jig 21 goes to a set of No. 4 bone rolls 23, thence to the condemned-coal elevator 24.

Exactly the same processes take place in the treatment of the stove coal from the second deck of shaker 18, except that a set of No. 6 rolls 25 is used to crush the bone. Chestnut from the shaker above mentioned is treated like the stove coal and the bone goes to the same set of No. 6 rolls 25.

The pea coal does not pass through a slater but goes directly to the pea jig 26, from which the cleaned coal passes directly to its pocket. On shaker 27, No. 1 buckwheat, rice, and barley coal are made. The buckwheat passes to the buckwheat jigs 28, the cleaned coal going to its pocket. Rice is likewise jigged. At this point the rice and barley coals from the pure side of the breaker join together and are sized on this deck of the shaker 27.

The barley coal is treated on Deister tables 29, after which pure barley is sent to the boiler house or to the market, as the case may be. The slush, which settles into the jig tub, is passed over a slush shaker 30 and the larger sizes removed and sent to the condemed-coal conveyor. The fine slush unites with that passing through the barley deck of shaker 27 and is sent to the slush dam. Lip screenings are taken by lip-screening elevators 32 to the clean egg shaker 10 for resizing.

As stated before, there are really two sides to this breaker—the pure-coal side and the bony-coal side. On both sides the treatment is wet. On the pure-coal side, the material receives no other treatment than the necessary breaking down and the sizing. Material sent to the bony-coal side, however, receives throughout a somewhat more elaborate treatment. In the first place, the coal, before it is jigged, is sized into flat and round particles, which simplifies the jigging. All sizes from egg down to rice are jigged, the barley coal being treated on Deister tables.

Rahn Breaker, the Lehigh Coal and Navigation Co.

This is the newest breaker of the Lehigh Coal and Navigation Co., and therefore embodies its latest ideas. A feature that differentiates it from other breakers is that the large rock is separated in what is known as a rock house and the coal that is taken into the breaker by the drag-line conveyor is comparatively clean.

Another feature is the breaker drive. Electric motors are used as the motive power but instead of installing motors to drive one or two pieces of machinery, only five motors are used in the whole building. One of these drives the conveying machinery, one the pumps, one the

shaker screens, one the rolls, and one the jigs. They are so disposed as to direction of drive that the vibration is reduced to a minimum. All the motors are controlled from a central switchboard. Another feature is the use of the cone separator for the finer material. Unfortunately this apparatus is only in the experimental stage and cannot be described.

The following is a description of the passage of the coal through the breaker; its progress can be followed readily on the flow sheet, Fig. 58. After being dumped, the coal passes over the bull shakers 2, on which

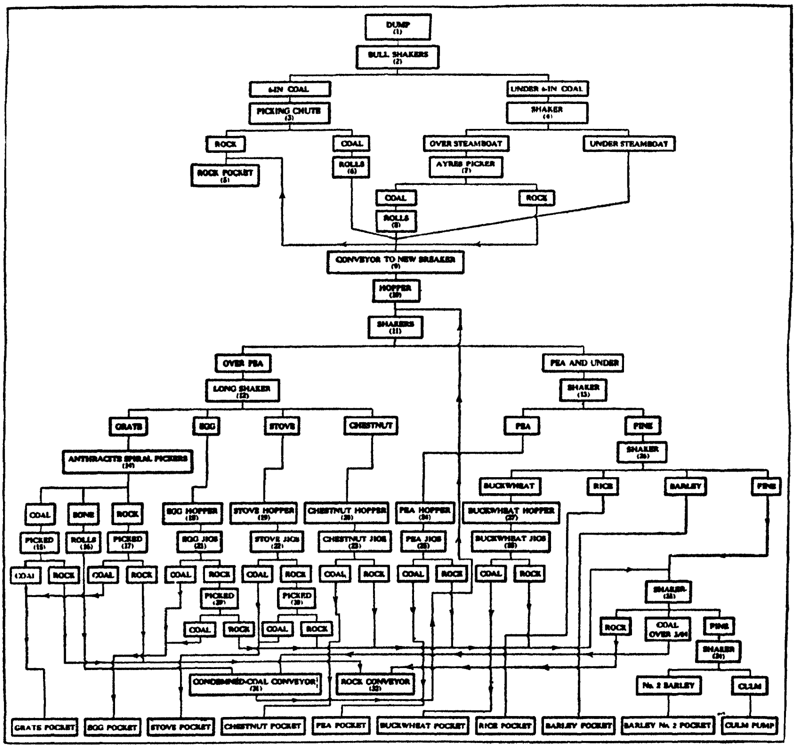

FIG. 58.—THE FLOW SHEET OF THE RAHN BREAKER.

two sizes are made; coal 6 in. (15 cm.) and larger passes over these shakers while that under 6 in. passes through them. The large coal then goes over a picking chute 3, where the rock is separated, being sent direct to the rock pocket 5, the coal passing through a set of rolls 6. Coal that passes through the bull shakers 2 goes over another set of shakers 4, on which two sizes are made—steamboat and larger, and coal smaller than steamboat. The larger size is picked on Ayres pickers and the coal is passed to a set of rolls 8, the rock being sent to the rock pocket 5. The

size under steamboat coal from shaker 4 goes to the conveyor leading to the new breaker, as does the coal from the rolls 6 and 8, this conveyor delivers the coal into a hopper 10 at the top of the new breaker, from which it is delivered by three feeders to three sets of shaking screens 11. On these shakers only two separations are made—over pea and pea and under.

This over-pea size passes directly from the shakers 11 to the long shakers 12. As is rather unusual in the anthracite field, these long shakers have only one deck each but make four separations, namely, grate, egg, stove, and chestnut. These shakers are 30 ft. (9 m.) long; on the first 12 ft. (3.7 m.) chestnut passes through, on the next 9 ft. (2.7 m.) stove coal is separated, and on the last 9 ft. egg goes through and coal of grate size passes over. This grate coal then goes to three anthracite spiral pickers 14, on which the coal, bone, and slate are separated. The coal is then hand-picked and passes to the grate pocket, the rock going to the rock conveyor. The bone coal goes to a set of rolls 16 thence to the condemned-coal conveyor 31. The rock from the spiral pickers 14 is hand-picked and the coal recovered is sent to the grate pocket, the rock being sent to the rock conveyor.

Egg coal from the long shakers 12 passes to a hopper 18 directly behind the three egg jigs 21. These are of the double type, and the coal is fed to them uniformly from the hopper, this uniformity in feed aiding the jigs to prepare the coal more thoroughly.

The jigs are of the Elmore type and make 100 strokes per minute. The coal that leaves them goes to the egg pocket, but the rock is hand-picked, the coal saved being thrown into the coal chute and passing with the balance of the coal to the pocket, the rock going to a shaking screen 33. Its subsequent treatment will be described later.

The stove and chestnut are prepared in the same manner as the egg, except that after the chestnut coal passes through the jigs the rock from the jigs is not hand-picked. Three double jigs handle the stove coal and three more the chestnut.

From shaker 11, the pea coal and smaller sizes pass to another shaker 13, which makes only one separation—pea and coals finer. The pea coal passes to the pea hopper 24, thence to the pea jig 25, where it receives exactly the same treatment as the chestnut coal.

The fine coal from shaker 13 is passed to a shaker 26, which has three decks and prepares four sizes of coal—buckwheat, rice, barley, and fine. Buckwheat coal is jigged just as is the pea and chestnut and receives the same subsequent preparation. Rice coal does not receive any further treatment and goes directly to the rice pocket. The barley coal, like rice, passes on without treatment. The fine coal from shaker 26 passes to shaker 33 and is fed on to its second deck; the rock from the jigs 21, 22, 25, and 28, is treated on the upper deck of the shaker.

Oversized rock goes to the rock conveyor while the undersize and the coal passes to the second deck of the shaker. All the particles over 3/64 in. (1.2 mm.) pass over this second deck to the condemned-coal conveyor, the finer particles passing through to another shaker 34, on which No. 2 barley is made. This size then goes to the No. 2 barley pocket, and the culm passes to the culm pump for delivery to the slush bank.

The coal delivered to the condemned-coal conveyor 31 is carried to the shaking screens 10 for treatment. The rock sent to the rock conveyor 32 is taken to the rock pocket and from that point hauled to the rock bank.

The breaker has a capacity of approximately 2,000 tons in 8 hr. The percentage of the different sizes of coal made, the amount of slate in the coal, and the percentage of the coal in the refuse slate are given in the accompanying table:

Size of Coal	Size Made, Per Cent.	Slate in Coal, Per Cent.	Coal in Slate Refuse, Per Cent.
Broken	3.8	...	2.0
Egg	7.8	2.0	2.0
Stove	9.7	3.5	2.8
Nut	17.2	5.0	3.1
Pea	10.6	8.0	3.5
Buckwheat	12.2	10.0	4.1
Rice	15.5		
Barley	20.2		
Barley No. 2	3.0	....	2.2

The domestic sizes of coal make 38.5 per cent. of the total produced and the fuel sizes amount to the large proportion of 61.5 per cent.

Three main and two minor drives are employed in this breaker. One of the main drives operates the machinery of the rock-separator house, another runs the jigs, and a third the shakers. A separate motor is used in each case, the jigs requiring a 400-hp. machine and the shakers one of 500 hp. Allis-Chalmers motors are used for both shaker and jig drives, being connected to the main shafts by 36-in. belts with pulleys on 50-ft. centers.

DISCUSSION

Sidney J. Jennings,* New York, N. Y.—We have been told by the newspapers and by some of the operators that anthracite is a luxury fuel. A visit to this district has convinced me that it is a very luxurious fuel. The operators here seem to have educated the housewives who are able to get anthracite to a degree of luxury that seems excessive. You can get a perfectly good fuel, even though it is a little bit smoky and dusty, for probably two-thirds of the cost of anthracite; but you cannot get the

* Consulting Engineer.

housewife to use that fuel if she can get the better one: Is it not, therefore, a part of the education in thrift not to prepare the anthracite to such an excessive degree of refinement? The sizing of the coal into so many sizes, as is now done, seems excessive.

The fact that the breakers work one shift adds, undoubtedly, to the capital expenditure and increases the cost of the fuel and puts it still further in the luxury class. I have roughly figured that the cost of the breakers, the model one that we have seen, is about $1.10 to $1.50 a ton of yearly capacity. Probably, by the addition of a storage bin at the breaker, which will receive coal on the day shift only, so that the breakers can be worked on three shifts, the capital expenditures can be reduced—certainly to about 60 or 75 cents a ton—by doubling or trebling the work any one breaker can do.

I suggested this to some of the anthracite operators, but they say that it is difficult to get men to work on the night shift. That problem has been faced by the workers in the mills for concentrating ores and has been solved, and I do not see why the anthracite breakers should be any more luxuriously looked after.

R. V. Norris,* Wilkes-Barre, Pa.—One point in connection with the seeming lack of energy in working day and night shifts, in many mines, is the development gangways. These are usually driven on two, and frequently on three, shifts and then they barely hold the 8-hr. output. It is a question of whether it is practicable to maintain development at a pace sufficient to work the chambers more than one shift and I think that is one of the underlying reasons why the coal mines throughout the world practically have developed on this basis.

Edwin Ludlow,* New York, N. Y.—This question has been carefully considered for we all realize the enormous capital expenditure in our breakers. The question of maintaining the breakers on a day shift, though, is one of labor, the costs of which are in excess of the interest on the capitalization. It has been found, in most of our experiments in trying to run two shifts (we never got to three), that it was cheaper to keep that breaker, at the time it was running, at a maximum output so that the hundred or more men employed in connection with it would be busy during that period of time.

One great difficulty is the fact that the contract miner is independent as to his hours and seldom works more than 4 or 5 hr. in any one shift. It takes therefore a large force of miners to keep the breakers going 8 hr. steadily. It requires three shifts on development in most mines having a large breaker, to keep that work far enough ahead. I doubt whether more than the one shift per day would be successful under present labor condi-

* Consulting Engineer.

tions. Most of us have had difficulty in maintaining the one shift to a mark above the minimum at which we know the breaker can be run economically. I doubt whether it would be practical at the present time to operate double shift. Our own experience has been that the greatest economy is to maintain the output of the breaker at its maximum on day shift.

C. T. Starr, Bethlehem, Pa.—How would Mr. Jennings limit the preparation, in sizes?

Sidney J. Jennings.—In the number of sizes.

H. G. Davis,* Wilkes-Barre, Pa.—The night shift question has been discussed for many years in the anthracite region. One of the coal companies, some years ago, tried to operate one of its breakers day and night in order to keep its mines in operation, having lost a breaker by fire, but the plan was not practicable.

English and German engineers who have visited the region invariably, on noting the tremendous outlay of capital connected with the development and operation of an anthracite mine, wonder why the breakers are not operated day and night so as to give a greater yield on the investment. This problem may yet be solved in a satisfactory manner since the machinery installed therein can be driven by separate electrical units.

I have at all times tried to eliminate the night shifts, for I have always thought that where men were employed on the night shift in order to get coal ready for the morning, somebody was not on his job. I have had no reason to change my mind up to this time. Of course, there are times when the development of a mine or a section justifies a night shift, when a number of chambers may be operated on the night shift in order to keep the transportation forces fully employed. In all cases, night shifts should be discouraged.

Our system of mining is entirely different from that in the bituminous field, and our method of preparing our product is much more complicated; but while I do not think that the breakers could be successfully kept in operation for the 24 hr., it may be possible, with the application of electrical units, to operate 16 hours.

Arthur Thacher, St. Louis, Mo.—Coming from the West, I am naturally in favor of night work. I know it is not carried on in the East, but in the West we find little difficulty with night operating. It seems a great economic waste to have all that capital expenditure working only a few hours. I recognize the fact that if the mines are operated for 24 hr. the storage of coal will be difficult. Coal is more bulky than ore. The question of bins is serious and if we cannot deliver the coal we might get into trouble.

* Consulting Engineer.

It has been said that the mines cannot be worked on three shifts because of contract labor; in my opinion contract labor is wrong and we will never reach our greatest success until we return to day wages; though I know that does not agree with most of the conceptions of the present day. Contract labor is one of the troubles today in England and Australia. I will go further and say that the day-wage system is practically applicable to bringing out the best results. I have obtained the best results from day pay, and some of the most remarkable records have been made in that way.

I do not mean that we can at once drop our contract work—you are controlled by the people of the community, by the customs and traditions of the village. I have not dropped it myself and will probably use it all my life. But we should recognize that it is not the best way to obtain the best results and we should be slow about trying to increase it.

F. B. Nold, Lansford, Pa.—The author referred briefly to experiments being carried on; has any one here had experience in the separation of coal in the breaker or mine by air, chiefly a preliminary cleaning before the main separation is made?

R. V. Norris.—There was some separation done by air by the Lehigh Coal and Navigation Co. some years ago; that was shown by a report published in connection with the summer meeting of 1911.

Edwin Ludlow.—The present plan is entirely different from that, as I understand it. This system is used at three or four places in the bituminous field, where it is doing satisfactory work. The idea is to carry the fine coal over a shaking screen through which a current of air is passed upward. This air current keeps the coal in agitation on the top of the screen while the slate drops through. I do not know that it has been tested in the anthracite region, as it is comparatively new.

R. V. Norris.—One point in regard to the question of night shifts is that all anthracite is mined by the contract system; that is, the miner contracts to mine the coal. If you want to work in a double shift you must either provide a double number of chambers, or else you must, in some way, force partnership agreements, so that one partner may work on one shift all the time. Both of these are somewhat impracticable.

Dever C. Ashmead (author's reply to discussion).—It has been suggested that anthracite mines and breakers should be operated on the three-shift basis. Three conditions will be considered and reasons given why operation of anthracite mines under these conditions would not be feasible.

Condition 1.—Whole mine to be operated on a three-shift basis.

Condition 2.—Present operating force to be divided into three parts, each part to operate a portion of the mine. Thus the first shift might operate the Red Ash, the second the Baltimore, and the third the Hillman bed.

Condition 3.—Operate the mine in one shift, as at present, and the breaker in three shifts.

The first condition could not be attempted with hope of success for frequently when the mines are being developed on the three-shift basis it is difficult to keep the development so far in advance of the producing forces, which work one shift, as to provide the latter with working; it would be, not only difficult, but impossible to do so if the coal-mining forces were trebled. Further, the men would continually quarrel, each claiming that a succeeding shift in the breast was loading out, and getting paid for, coal dislodged by the preceding shift and not leaving the shift that followed an equal amount of coal to load out.

The introduction of the three-shift system would probably necessitate the introduction of contract work or partnership on a large scale. One man would take the contract for the chamber, or breast, and would then hire men to work it for the three shifts. For years the miners have been trying to have the contract system ended, and they would resent any readjustment that would increase the prevalence of the practice that they have unsparingly condemned.

In the second condition, the mine will be operated on three shifts without increasing the force of miners, the work being spread over three shifts instead of one and each place being worked by the same number of men as at present. This overcomes the difficulty of providing development and also gives possession of one place to a miner and his loader, eliminating disagreements. But it will cause the operator much trouble, for the miners in the anthracite region, as a whole, are not used to working except on the day shift, therefore to force two-thirds of the men to work other shifts would immediately cause unrest and strikes.

Furthermore, at many mines there is only one shaft, so that working three shifts would mean that a full force of shaft men would be required to handle the coal; in other words, the labor cost of hoisting would be trebled. Besides, the underground-haulage cost would be increased, the present arrangements underground being such that the coal often must be transported from one bed to another, and the same haulage system is practically used for a number of beds. Therefore this system would be kept in operation for all three shifts using practically a full working force for each 8-hr. period. Larger gangs of repairmen would have to be employed. The ordinary day force might be decreased but the number of repairmen who would have to be employed for the other two shifts would be great enough possibly to double the present forces. The whole cost of mining would be increased, and it is extremely doubtful if the saving resulting in

the smaller amount of capital required would begin to pay for the increased cost of operation.

The third condition is likewise impracticable. If the mine worked only a third as long as the breaker, it would be necessary to provide storage for two-thirds of the coal produced. For instance, if a colliery produces 6,000 tons of coal a day, the breaker will have a capacity of 2,000 tons, therefore storage must be provided for 4,000 tons. The storage pockets must be of fireproof construction, and every facility for the quick handling of this coal must be provided. It is questionable therefore, whether the decreased cost of the breaker would pay for the construction of the storage pockets. In addition, while the 2,000-ton breaker will prepare only one-third as much coal as the 6,000-ton breaker, the cost will not be reduced in proportion; the saving in the cost of construction might be only from one-third to one-half. Also the coal must be taken out of storage so that breakage will be increased and a decrease in the prepared sizes will result in lowering the return on the coal.

The cost of supervision would be increased by three-shift operation and the coal would have to be inspected over a period three times as long as at present. The capacity of the breaker per shift would be greatly decreased, but the force operating it would not be reduced in proportion, as a 2,000-ton breaker would require the employment of more than one-third of the men demanded by one of 6,000-ton capacity. These last reasons apply, of course, whenever the breaker is run on three shifts whether the mine works steadily or only for 8 hours.

The margin of profit in the anthracite region is so small that it would be impossible to operate the coal mines at any increased cost. The introduction of any such systems would upset the whole labor situation, and the losses that would arise from a strike of a few months duration would exceed all profits that might come from such a change. In my opinion no profit would be derived from such a change; in fact, the introduction of such a system would increase the cost of operation.

Slush Problem in Anthracite Preparation

BY JOHN GRIFFEN,* SCRANTON, PA.

(Wilkes-Barre Meeting, September, 1921)

THE modern anthracite breaker or washery uses almost exclusively a wet method of preparation, which requires, roughly, 1 gal. of water per minute per ton of production per day. The entire anthracite industry uses about 320,000 gal. per min. of water for this purpose or 800,000 tons of water per day. As this water leaves the breakers, it contains fine solids—coal, slate, pyrite, and clay—and is then called silt or slush; as slush is the term most commonly used, it will be employed throughout this paper. The solid content of the slush will be referred to as solids.

In the earlier days of anthracite mining, little coal was washed, less crushing was employed, and virgin coal was mined exclusively. As a result, the slush problem was not acute because of the relatively small quantity and the coarseness of the solids. Such slush as was produced could usually be easily impounded and retained or could be discharged into streams without any appreciable pollution being apparent.

The character of the fine waste from the breakers changed materially as its quantity increased until now about 40,000 tons of slush solids are produced daily. Second mining and robbing operations materially increased the quantity of fine solids delivered to the breaker in the mine car. The demand for chestnut, stove and egg sizes, to the exclusion of grate, steamboat, and lump sizes, requires finer crushing of the mine-run coal with a consequent increase of fine solids in slush. The use of rice and barley sizes removed a considerable tonnage of coarser solids from the slush but left great quantities of fine solids difficult to retain completely, and to store, and long considered of no possible fuel value.

Despite the efforts of the coal operators, the slush solids have found their way into the streams causing, in some cases, serious pollution. The Water Supply Commission of Pennsylvania published, in 1916, a report on Culm in the Streams of the Anthracite Region,[1] from which the following is taken:

> About 40,000,000,000 gal. of water carrying 10,000,000 tons of fine culm are discharged into the water-courses direct, flushed into the mines, or disposed of by various means on the surface. The extent to which the very small sizes of anthracite have been deposited in the rivers draining the coal fields is made evident by the fact that over a quarter of a million tons are recovered annually from the river beds by coal-washing operations.

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* Manager, Anthracite Territory, The Dorr Co.

[1] Water Resources Inventory Report, part X.

The contamination of the streams has been in progress for more than 50 years and it is estimated that there are now 660 miles of creeks and small streams, which should be available for water supply but which are rendered useless for domestic and manufacturing purposes by the culm and sulfur water from the mines.

Since 1914, the quantity of solids reaching the streams has probably not increased, but it has changed in character. Due to the war demand for steam sizes, less of these sizes was lost from the breakers and many of the banks that could be washed into the streams by heavy rainfall and freshets were prepared for use and shipped. As a result, it is probable that far less coal of the buckwheat sizes is now finding its way into the streams.

The recovery of coal from the rivers increased enormously during the war period but the coal was largely obtained from deposits formed by the waste of earlier years. W. C. Webbert,[2] in an address before the Engineers' Club of Philadelphia, said: "The total output of coal for the Lehigh River for 1919 was 120,000 tons; adding to this 235,000 tons for the Schuylkill and 1,580,000 tons for the Susquehanna and its tributaries, the total output of river coal in eastern Pennsylvania for the year 1919 can be estimated at 1,935,000 tons. Approximately, the same amount of coal was reclaimed in 1918 but prior to that year the output was much less."

Character of Slush

Slush, as now discharged from the breakers, contains practically no solids larger than 3/32 in. (2.3 mm.) diameter and often passes through a round mesh of 1/16 or 3/64 in. (1.5 or 1.19 mm.) diameter. It carries from 4 to 15 per cent. of solids by weight, and in isolated cases an even higher percentage. The solids will range, in size, down to particles that are colloidal and, under certain conditions, settle very slowly.

Analyses of the various sizes of solids in slush generally show that the ash content, that is impurities, increases with decrease in size. The following typical screen analysis illustrates this condition:

	Per Cent. Sizes		Per Cent Ash	
	Each Size	Cumulative	Each Size	Cumulative
On 3/32-in. round screen	5.3	5.3	27.38	27.38
Through-3/32 in. round screen on 20 mesh	21.7	27.0	30.26	29.70
Through 20 mesh on 35 mesh	15.0	42 0	34.40	31 40
Through 35 mesh on 48 mesh	6.4	48.4	34.80	31.80
Through 48 mesh on 65 mesh	4.7	53.1	34.10	32.00
Through 65 mesh on 100 mesh	5.1	58.2	36.10	32.40
Through 100 mesh on 200 mesh	6.8	65.0	38.84	33.10
Through 200 mesh	35.0	100.0	59.40	42.30
	100.0			

[2] Bureau of Topographic and Geological Survey, State of Pennsylvania.

The granular solids that will remain on a 200-mesh screen are reasonably low in ash and consist of grains of fairly pure coal mixed with grains of slate, sand, pyrite, and, occasionally, calcite and gypsum. The slimes, which will pass a 200-mesh screen, consist largely of fine slate and fine clay and show a high ash content.

The quantity of solids in slush and their character varies greatly from breaker to breaker. A fairly extensive investigation permits the following generalizations:

1. Steep-pitch mining produces a greater quantity of slush solids and generally causes a large quantity of slimes, as well as considerable impurities in the plus 200-mesh solids.
2. Second mining and robbing operations, where crushing has taken place, have the same effect as steep-pitch mining and generally produce a colloidal slime, which settles with difficulty.
3. A coal of soft and friable nature increases the amount of solids, which are relatively low in ash, even including the slimes.

Quantity of Slush Produced

An accurate determination of the tonnage of slush solids made annually is quite difficult but a close approximation can be obtained by careful tests at breakers representing the various fields throughout the region and applying these figures to the total production of the fields.

In figuring the tonnage of recoverable coal, no solids passing through 200 mesh are included and the amount of refuse larger than 200 mesh that must be removed to reduce the ash content to 15 per cent. has been deducted.

A summary of the determinations of the three fields shows the following amounts of solids in the slush, expressed as per cent. of shipments:

	Wyoming Field, Per Cent.	Lehigh Field, Per Cent.	Schuylkill Field, Per Cent.
Total solids	7.2	22.4	22.3
Recoverable coal with 15 per cent. ash	3.5	9.5	8.8

The Anthracite Bureau of Information states that the shipments for the past three years were as follows:

Year	Wyoming Field, Long Tons	Lehigh Field, Long Tons	Schuylkill Field, Long Tons	Total, Long Tons
1918	42,382,793	11,511,760	22,755,365	76,649,918
1919	36,689,313	10,266,479	19,899,519	66,855,311
1920	37,249,303	9,860,611	21,517,211	68,627,125
Average, 1919–1920	36,969,308	10,063,545	20,708,365	67,741,218

The shipments during 1918 were at an unusually high rate, while 1919 and 1920 are more nearly representative.

Applying the field averages for total solids and recoverable coal to the average shipments gives the following annual tonnages in the slush:

	Wyoming Field, Long Tons	Lehigh Field, Long Tons	Schuylkill Field, Long Tons	Total, Long Tons
Total solids	2,661,000	2,254,000	4,618,000	9,533,000
Recoverable coal with 15 per cent. ash	1,294,000	956,000	1,822,000	4,072,000

This indicates that 4,000,000 tons of coal can be recovered from the slush at present produced, leaving 5,500,000 tons of solids, a large part of which is finer than 200 mesh, which must be retained in some efficient way if serious pollution is to be prevented. It must be noted that the trend in mining anthracite indicates that the tonnage of solids in slush and recoverable therefrom will increase rather than decrease, for more and more robbing is being resorted to and much of the reserves from which future production must come lie in the Schuylkill field.

Methods of Recovering Slush Solids

It is probable that the first efforts to retain the solids in slush consisted in allowing the solids to settle back of retaining dams, and were to prevent stream pollution. As the size of the dams increased, it became necessary to find other places for storage or a method that would hold a greater tonnage on a given ground area. Slush is now disposed of in the following ways: Run to slush dams; used for hydraulic mine filling; delivered to settling tanks of various types where the solids are removed and stocked, burned in mine-boiler plants, or shipped.

Much of the slush produced is still delivered to settling dams, which effect a more or less complete removal of solids. These dams are constructed in two ways: Either the slush is impounded to form a pond and the clarified water is allowed to overflow the top; or the retaining dam is constructed of porous material and the water is filtered in its passage through the dam. Either type, if given proper care, can be made very efficient and an almost complete removal of solids obtained so that the clarified water contains less than 0.1 per cent. solids.

Where the slush is impounded and clarified by sedimentation, a large dam must be constructed so as to give plenty of settling area and a depth of several feet of water should be maintained. This requires the labor of several men and often large quantities of lumber. Unless the men are carefully watched, the dam will not be properly maintained and only the coarsest solids will be retained. When the water is several feet deep, it is difficult to keep the slush and water from breaking through unless the dam is heavily reinforced with breaker slate, mine rock, ash, or lumber. At many collieries, slush dams must be located on territory that is broken and caved from mining operations and much of the water,

and often slush solids, finds its way into the workings and seriously increases the pumping load.

Slush dams that clarify by filtering are usually constructed of a core of mine rock and breaker slate. Boiler ashes are dumped along the inside of this wall and act as a filtering medium. It is often difficult to seal all the large passageways so that slush solids will not pass. After these passages are sealed, however, the dam will deliver fairly clean water. The ashes finally become clogged with slush solids, so that the filtering operation stops; the dam walls must then be raised to offer new ashes for the filtering process.

Slush dams afford a fairly low-cost method of retaining slush solids to prevent pollution where the slush can be run to the dam by gravity. The retained solids are invariably so permeated with slimes and fireclay that they cannot be utilized for fuel without further preparation.

Hydraulic Mine Filling with Slush

Slush was used as early as 1884 to extinguish a serious mine fire. It was early realized that such flushing, in addition to serving other purposes, gave a possible means of disposing of slush and preventing stream pollution. It is seldom, however, that mine flushing of slush can be applied solely for the prevention of pollution, as the cost generally is high. If, however, fires are to be extinguished, surface supported, or filling required to enable further extraction of pillar coal, mine flushing of slush may be economic. Tests made by Prof. F. B. McKibben and W. H. Conklin,[3] at the Fritz Engineering Laboratory of Lehigh University, show that slush solids, when confined so as to prevent lateral expansion, will support heavy pressures per square foot. The use of slush for hydraulic mine filling creates many problems where the coal measures are steeply pitching, owing to the difficulty of holding the solids in place while the water is drained away.

Charles Enzian[4] gives the cost of hydraulic mine filling with slush as from 9 to 33 c. per cu. yd. when operating at a rate of at least 400 cu. yd. daily. These costs are based on prices in force during 1911 and 1912 and should be corrected for present-day conditions. They do not include anything for the value of the coal in the slush. When there is a method of utilizing the coal in the slush that will return a greater revenue than the mine coal won by flushing with slush, this method of disposal will cease in all but exceptional cases.

Slush Settling Tanks

Settling tanks of various types were developed to recover as much of the solids as possible in a relatively dry condition, so that a large

[3] William Griffith and E. T. Connor, Bureau of Mines *Bull.* 25 (1912) 55.

[4] Bureau of Mines, *Bull.* 60.

tonnage could be stocked on limited areas. Such tanks also enabled the recovery of a crude product for shipment, for which there has been but small demand for certain special uses—mainly in the metallurgical industries.

Intermittent Type

The earliest settling tanks installed work intermittently and consist of a series of hoppers or tanks with gates at the bottom. The slush is delivered into one tank until it is filled with solids and is then diverted into a second, while the solids in the first are discharged from the bottom into cars or a conveyor line by which they are delivered to the bank. These tanks remove solids plus 100 mesh rather completely, if made large enough and given proper attention. The solids recovered contain layers of fireclay and slime, which render them unfit for boiler fuel. One man's time is required to regulate the flow of slush and to discharge each tank when filled. No power is taken for tank operation, but installations can be made only where the necessary headroom is available. The cost of repairs on these tanks is low as no machinery is employed, but due to the attention required, these tanks have given way to tanks where the operation is continuous.

Bucket Elevator Type

Various arrangements of bucket line, feed, and overflow are used. In some cases buckets with perforated sides are employed while in others a solid bucket is used; but the bucket is slightly tilted after leaving the water in order to decant off the excess water. Unless the bucket line is quite large and is operated at a low speed, only the coarsest solids will be removed. Many of these tanks lose some solids larger than $\frac{3}{64}$ in. mesh, while at the same time some solids as fine as 100 mesh are removed. Due to the agitation of the bucket line, little slime and fireclay are found with the recovered solids, which can be used as fuel in properly designed boiler plants. The attention required is confined mostly to lubrication and repairs, but the construction costs are fairly high and repairs to the elevator are difficult and expensive. As the buckets and chain are submerged in the slush for much of the time, wear is heavy. At one colliery, where the water is acid, buckets last about 6 months and the chain a year.

Drag Flight Conveyor Type

A conveyor is installed in a shallow horizontal tank of some length. At the discharge end, the conveyor is carried out of the tank on a pitch that allows some drainage of the removed solids and returns over the top of the tank. The conveyor flight is operated at a speed not greater than 50 ft. per min.; the sectional area of the tank should be such that the speed of the slush through the tank will be the same as the speed of the

flight; so that the agitation caused by the flight will be reduced to a minimum. The slush is fed near the tail wheel of the conveyor and the overflow is taken off toward the discharge end.

Some of these tanks make a good recovery of solids. One tank, with the horizontal part of the conveyor 60 ft. in length, using 6 by 18 in. (15 by 45 cm.) flights, spaced 18 in. apart and running 50 ft. per min. in a tank 2 ft. 8 in. inside width, handled 800 gal. per min. of slush through 1/16 in. round mesh, carrying about 13.5 per cent. by weight of solids, and reduced the solids in the overflow to 1.2 per cent., or a 91 per cent. recovery of solids. No data are available as to the size of the solids in overflow and recovered product, but it is probable that practically all solids larger than 200 mesh were recovered. This is the best performance record obtainable and tests were made shortly after the settling tank was first operated. Experience shows that the efficiency decreases after these tanks have been in service some time. The efficiency of this tank falls off if the slush feed is subject to heavy rushes, such as occur when a number of jigs in the breaker are slushed simultaneously.

The coal recovered usually contains all the plus 100-mesh solids in the slush, but it also contains sufficient finer solids to make its utilization uneconomic without further preparation. It is usual to assign one man to the operation of these tanks but he generally has time for additional duties.

The construction costs are moderate; one tank designed to handle 2000 gal. per min. of slush cost $9000, including the cost of pipe line to tank and stocking conveyor for the product. An installation erected during 1920–21, consisting of 8 by 18 in. (20 by 45 cm.) flights spaced 18 in. apart, 120 ft. long center to center of sprockets, in a tank 4 ft. wide by 4 ft. deep by 100 ft. long, with a stacking conveyor 65 ft. center to center on a 6 in. pitch and the same flights spaced 3 ft. apart, cost $10,253.

Repairs on this type of tank cost less than on the bucket elevator type but they amount to a considerable sum where acid water is employed, owing to the wear on chain and flights.

New Types of Equipment

During the past two years several slush recovery plants, containing Dorr thickeners and classifiers, have been installed; they have shown the following advantages:

1. Operation can be controlled so as to produce a product of given specification.

2. Recovery of over 90 per cent. of the slush solids can be obtained or any lower percentage as desired.

3. Maintenance costs are reduced to a minimum.

4. Power and attendance costs are extremely low

5. Additional equipment can be conveniently included in the plant to remove slate, sand, and other impurities from the recovered coal, giving a product analyzing 15 per cent. ash or less.

The equipment operates continuously and is so designed that no bearings are submerged in the material being treated. Because of the prevalence of acid water, wooden-stave tanks are used for thickeners up to 40 ft. in diameter, while for the larger sizes concrete tanks or tanks with wooden-stave sides and clay bottom built in the ground are used.

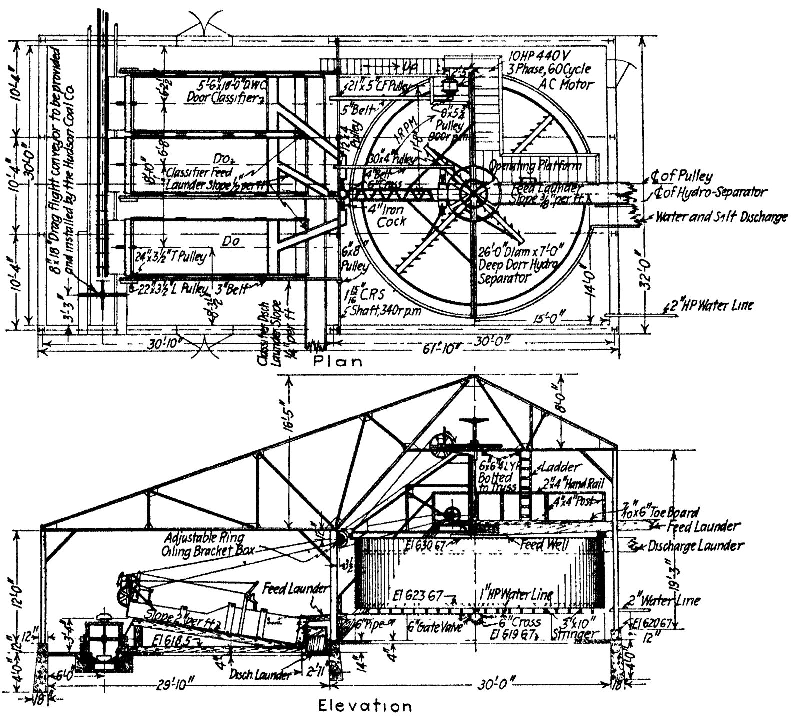


FIG. 1.—PLAN AND ELEVATION OF FINE-COAL RECOVERY PLANT.

The plows rotate so slowly that no swirl or agitation is produced. Sufficient tank area can be provided to cause the settlement of all suspended solids, giving a practically clear overflow. In case only the coarser solids are to be removed, a tank of smaller area is provided, so that the finer solids are carried into the overflow with the water. A thickener operating under this condition is termed a hydroseparator. Rough separations can be made at any desired mesh.

On anthracite slush, the classifiers can be operated to recover a product that is practically all larger than 48 mesh; or any part of the solids

between 48 mesh and 200 mesh can be included in the recovered product. Where solids smaller than 200 mesh must be completely recovered, a thickener is required.

Slush treatment is generally practiced to obtain the recovery of fine coal, prevent pollution, or to recover clarified water for breaker use.

In the recovery of fine coal, all the solids are sufficiently low in ash for use at the mines or for shipment; the slimes are too high in ash for present use; or the granular solids also carry so much impurities that removal of them is necessary to produce a useable product.

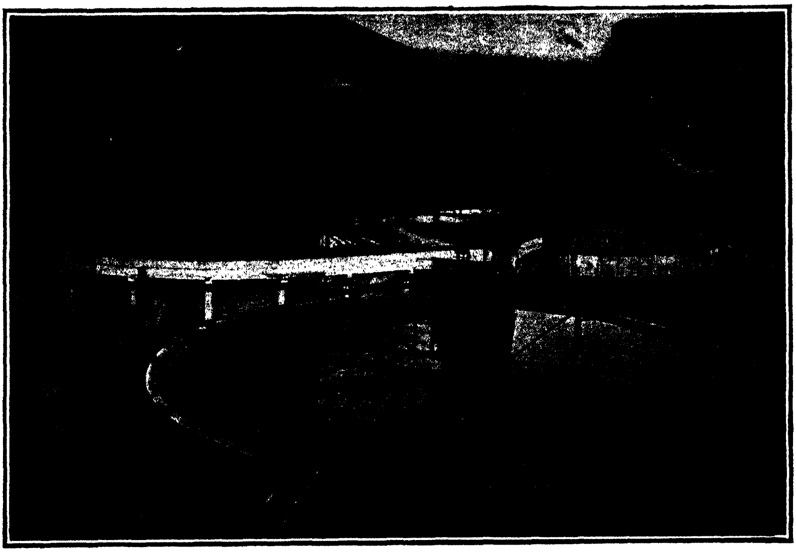


FIG. 2.—DORR THICKENERS HANDLING OVER 6000 GAL. PER MIN. OF WASTE WATER FROM A BITUMINOUS COAL WASHERY.

No installations of Dorr equipment have yet been put in operation in the anthracite fields to meet the first condition, as this condition is but seldom found except in the Lykens Valley district.

Several plants have been installed to recover the granular coal and more are in course of erection. The first plant of any size for this purpose was installed in the Wyoming field at a breaker producing 5000 to 6000 tons per day, and was designed to recover the solids larger than 60 mesh. A Dorr hydroseparator 26 ft. in diameter by 8 ft. deep and three Dorr classifiers were installed. A plan and an elevation of the installation are shown in Fig. 1. The slush amounts to about 4000 gal. per min., contains about 5 per cent. total solids, and is made through a $\frac{3}{64}$-in. round-mesh screen. The plus 60-mesh solids form about 40 per cent.

of the total solids, while the solids between 60 and 100 mesh amount to 20 per cent. additional. This installation is recovering the plus 60 mesh coal, with 15 to 20 per cent. undersize and with 40 per cent. moisture. On delivery to the stock pile, the product quickly drains to 18 per cent. moisture. With breaker shipments averaging 5000 tons per day, 200 dry tons per day are recovered from the slush. On a number of days, the breaker has shipped well over 6000 tons without taxing the Dorr equipment. The hydroseparator easily takes care of the material when a number of jigs in the breaker are slushed out. At such times, the slush will amount to 6000 gal. per min. for several minutes.

The installation was made in the summer of 1919 and cost as follows, exclusive of stacking conveyor line:

Foundations	$ 1,800.00
Equipment	9,225.32
Erection of equipment	746.92
Building erected	4,965.76
Total cost.	$16,738.00

The plant is operated by one man and a 10-hp. motor. During 2 years of operation, no parts of the hydroseparator or classifiers have required replacement and now show little signs of wear. Total operating costs, including 10 per cent. of plant cost for fixed charges and 10 per cent. for amortization, have been from 9 to 10 c. per dry ton of product.

A slush plant to recover a low-ash granular coal, now being installed in the Wyoming field, consists of a 32 ft. diameter by 8 ft. deep Dorr hydroseparator, eight Deister-Overstrom coal-washing tables, and four Dorr classifiers. It is designed to treat 4000 gal. per min. of slush made through 3/64-in. round-mesh screen and recover the plus 200 mesh coal with an ash content of 15 per cent or less. The breaker ships from 4000 to 5000 tons of coal per day.

The slush will be delivered to the Dorr hydroseparator, where the bulk of the water and solids minus 200 mesh will be separated and discharged to waste in the overflow All the plus 200 mesh solids will flow by gravity from the bottom of the hydroseparator to a distributing launder for feed to the eight tables. On the tables, the refuse will be removed and the washed coal will flow by gravity with the water used on the tables to four Dorr classifiers, where the coal will be recovered and dewatered. It is expected that about 200 dry tons per day of coal between 3/64-in. mesh and 200 mesh analyzing less than 15 per cent. ash will be recovered.

This equipment is being installed in a separate building of steel construction and, at present, construction costs are not available. The power consumption will amount to about 25 hp. Probably one man, and

certainly not more than two, will operate the plant. Operating costs, excluding fixed charges and amortization, should amount to less than 10 c. per ton of product.

Dorr Plants to Prevent Pollution

Plants for prevention of stream pollution may be of various types, depending on the amount and size of solids that must be retained. One plant installed, in 1920, in the Lehigh field consists of a 30 ft. (9 m.) diameter by 7 ft. deep Dorr hydroseparator and two Dorr classifiers. It is recovering the granular solids from 2200 gal. per min. of slush made through a 1/16-in. (1.5 mm.) round-mesh screen. The purpose of the plant is to remove all the solids that would block the stream into which the breaker slush water must discharge.

Prior to the installation of Dorr equipment, two settling tanks of the bucket elevator and drag flight conveyor type had been used in series but the stream could not be kept free of slush solids. After the Dorr plant had been in operation only a few weeks, the stream bed for several miles below the breaker had become freed from the solids already deposited, and after 7 months operation no signs of deposit in the stream are apparent

The construction costs of this installation, which was placed in a separate building 1/2 mile below the breaker, in order to get plenty of stacking room, were as follows:

Foundations	$ 530.00
Dorr equipment	6,856.00
Building	2,602.00
Wooden pipe line to plant	2,800.00
Stacking conveyor	4,576.00
Electrical equipment, including transmission line and transformers	2,620.00
Labor	6,479.00
Total	$26,463.00

The construction cost of this plant is rather high because of the wooden pipe line, transmission line, and substation required. In many cases the equipment could be installed in the breaker, which would have saved at least $10,000.

This plant is operated by two men, who also look after the disposition of the product on the stock pile, to which it is delivered by an 8 by 18-in. conveyor, 120 ft. long. A 15-hp. motor drives the Dorr equipment and a 30-hp. motor the stacking conveyor. Although the slush water comes from the mines and is quite acid, the maintenance costs have been quite low. One casting on the hydroseparator, weighing not over 200 lb., wore out in 4 months and was replaced by a bronze casting; otherwise the

equipment shows no signs of wear from the acid water and abrasive action of the solids. It is too early to figure accurate operating costs but operation to date indicate costs, exclusive of fixed charges and amortization, about 7 c. per ton of recovered coal. Based on breaker shipments, the cost of preventing pollution is slightly under 1 c. per ton shipped, if the recovered coal is considered of no value. If the recovered coal is valued at 18 c. per ton, the cost of preventing pollution is paid by the value of the coal.

A Dorr plant of a different type is planned for a breaker in the Schuylkill field; it will probably be erected during the summer of 1921. At this colliery the question of pollution is rather serious and an almost complete removal of solids is necessary. A Dorr thickener 90 ft. in diameter will be installed in a tank with wooden stave sides and clay bottom. The feed will amount to 2200 gal. per min. of slush made through a 1/16-in. round-mesh screen and containing about 320 tons of solids per 8-hr. day. The thickener should remove about 300 tons per day, leaving nothing in the water but the finest slimes. The recovered solids, with about 50 per cent. moisture, will be delivered to a conveyor line installed in a concrete tunnel under the thickener for delivery to the storage pile. The recovery of total solids in the slush will amount to more than 93 per cent.

This thickener will produce a clarified water that will be suitable for all washing processes in the breaker. Fig. 2 shows three thickeners installed in the bituminous field for a similar purpose.

Utilization of Fine Coal Recovered from Breaker Slush

It is our opinion that all anthracite shipped should, and eventually will be, converted into a form that can be used as a domestic fuel. The gradual exhaustion of the anthracite reserve tonnage and the increased cost of extraction will gradually force the conversion of the buckwheat sizes and the coal now lost in the slush into some form of fuel satisfactory for domestic use. The time will come when practically the only anthracite tonnage used for generation of power will be that required for mine fuel. During the past few years, considerable work has been done toward the utilization of the finer sizes of anthracite. It is the purpose of this paper simply to summarize the accomplishments to date.

Briquetting

Considerable progress has been made in the briquetting of anthracite; in 1920, nine plants were in operation and produced 330,125 short tons. This production was 50 per cent. higher than the output of any previous year. According to reports to the government, the average price received, during 1920, for all briquettes, whether of anthracite or other coals, was $7.50 per short ton.

Operating costs, during 1918, for one plant of 50,000 to 75,000 tons annual capacity, using the Dutch oil process, were as follows per gross ton:

Binder	$1.25
Slush	1.00
Superintendence and labor	0.40
Power, light, heat, and water	0.15
Supplies	0.05
Maintenance	0.05
Interest on investment	0.10
Depreciation	0.20
Insurance, compensation, and taxes	0.05
Royalty	0.10
Total	$3.35

This amounts to $3.00 per short ton.

As 4,000,000 gross tons of fine coal carrying under 15 per cent. ash can be recovered from slush at an average cost of 30 c. per ton and delivered to the briquetting plant and dried for a total of 75 c. per ton, the above costs indicate an attractive margin of profit, provided a market for the tonnage can be obtained at prices approaching those of stove and chestnut. So far trouble has been experienced in producing an anthracite briquette that in all respects can be substituted for anthracite. The main difficulty has been to produce a briquette that could be fired like anthracite without producing smoke, soot and odor. Conversion of the 4,000,000 tons of recoverable low ash coal in the slush into briquettes would add approximately 9 per cent. to present shipment of domestic anthracite.

Anthracoal

During the last few years, a new fuel has[5] been developed by heating, in a coking oven, a mixture of fine anthracite and coal tar pitch, or similar bituminous materials. This fuel has much the appearance of coke, but greater hardness, density and strength. Its volatile matter averages from 2.5 to 3.0 per cent. and it exhibits all the burning qualities of anthracite. Several carloads have been manufactured, shipped and burned as a domestic fuel with most satisfactory results. Tests made on a semi-commerical scale of operation, show that the cost of manufacture will not exceed that of furnace coke.

As the fine anthracite used as raw material can be cleaned so as to show an ash content less than 15 per cent., anthracoal can easily be produced carrying less ash than average domestic anthracite. As the domestic user is required to make no changes from his method of burning anthracite, one of the most serious marketing obstacles will not arise and prices should be received for anthracoal at least equal to those paid for domestic sizes of anthracite.

[5] Donald Markle: Anthracoal: A New Domestic and Metallurgical Fuel. See page 535.

Utilization of Fine Coal as a Steam Fuel

Many of the larger anthracite companies have done considerable experimental work in developing the most economic method of burning anthracite smaller than barley size. These experiments have been conducted along the following lines: (1) Hand firing, (2) firing on Coxe stokers, (3) hand and stoker firing with mixtures of bituminous coal, (4) pulverizing and burning in suspension.

Hand Firing.—One of the large companies in the Wyoming field has experimented extensively with hand firing of coal from slush but has abandoned the idea in favor of stoker firing because of the labor problem involved. Stationary grates of the pinhole type and about 4 per cent. space were used.

Mixing of coal from slush with barley size reduces the labor problem somewhat and increases the capacity and economy of the boiler. The following tests were run on a mixture of one-third barley and two-thirds coal minus 3/64-in. mesh. The size of fuel fired was as follows:

	Per Cent.
On 3/16 in. round mesh	2
On 3/32 in. round mesh	17
On 1/16 in. round mesh	25
On 3/64 in. round mesh	33
Through 3/64 in. round mesh	67

	Boiler No. 7	Boiler No. 8
Horsepower developed	310.7	313.9
Per cent. rating developed	115 0	116.3
Actual evaporation per pound dry coal fired, pounds	3 95	4.09
Equivalent evaporation per pound dry coal fired, pounds.	4.20	4.35
Heating value per pound dry coal, B.t.u.	12,133 0	12,133 0

The results are about 60 per cent. of the performance usual on similar boilers burning barley size hand fired.

Stoker Firing.—A fairly large tonnage of coal finer than barley is used as fuel for Coxe stoker-fired boilers in the Wyoming field. One of the large companies in the field uses coal recovered from slush to the extent of nearly 50 per cent. of its boiler-plant fuel. The character of this slush, which it terms No. 2 barley, is as follows:

Size Test	Per Cent.	Analysis as Fired	Per Cent.
On 10 mesh (3/32 in. round)	18.30	Moisture	10.0
On 20 mesh (3/64 in. round)	69.50	Volatile combustible	5.5
On 30 mesh	85.20	Fixed carbon	67.2
On 60 mesh	94.95	Ash	17.3
On 100 mesh	98.00		100.0
Through 100 mesh	2.00	Heating value	11,010 B.t.u.

During the year 1920, 394,000 tons of No. 2 barley were used by this company for mine fuel. Two of this company's plants use No. 2 barley exclusively and their average performance for the year 1920 was as follows:

	Plant A Pounds	Plant B Pounds
Actual evaporation per pound coal, as fired	4.74	4.32
Equivalent evaporation per pound coal, as fired	5.36	
Equivalent evaporation per pound dry coal	5.95	

These boiler plants are operated with ease at 150 per cent. rating for the day period of 8 hours.

This company used but 8.35 per cent. of its total production in 1920 for mine fuel. Coal recovered from slush, No. 2 barley, amounted to 4.10 per cent., leaving a net consumption of commercial sizes for mine fuel of only 4.25 per cent. of total production.

Recently tests were made on Coxe stoker-fired two-drum Sterling boilers using fuel recovered from the slush of a breaker situated in the Wyoming field. The fuel was recovered from the slush by Dorr equipment and ran in size from 3/64-in. round to 100 mesh, or considerably finer in size than No. 2 barley. The results of tests were as follows:

Builder's rating developed, per cent	152 6
Equivalent evaporation per pound of dry coal, pounds	4.74
Moisture in fuel as fired, per cent	12.2
Ash in dry fuel, average, per cent	28 80
Heating value of dry fuel, average, B.t.u	10,224
Overall efficiency, per cent	44 8

Tests with the same size coal that had been cleaned on a Deister-Overstrom concentrating table to 17.4 per cent. ash, failed as the moisture content was too high for proper combustion. The high moisture can be eliminated when the concentrated coal should show better performance than the high ash coal. The tests on Dorr coal were only preliminary. Tests run during 1918 on a 463-hp. four-drum Sterling boiler equipped with Coxe stoker using bird's-eye and a mixture of half bird's-eye and half Dorr coal gave the following results:

	Half Bird's-eye and Half Dorr Coal	Bird's-eye
Builder's rating developed, per cent	139.4	180.1
Equivalent evaporation per pound dry coal, pounds	6.19	7.61
Overall efficiency, per cent	52.16	61.83
Efficiency of boiler and furnace, per cent	62.74	67.37
Efficiency of grate, per cent	83.14	91.87
Combustible in ashes and refuse, per cent	17.51	21.47
Coal size test, on 5/16 in. round screen, per cent	2.8	6.6
on 3/64 in. round screen, per cent	51.1	92.8
through 3/64 in. round screen, per cent	48.9	7.2
Moisture in coal as fired, per cent	9.60	9.12
Ash in dry coal, per cent	21.12	18.26

The quality of the Dorr coal used in the mixture was as follows:

	Per Cent.
On $\frac{3}{32}$ in. round mesh	2.6
On $\frac{3}{64}$ in. round mesh	9.4
On 60 mesh	69.2
On 200 mesh	94.8
Through 200 mesh	5.2
Ash in dry coal	25.0

Slush Coal Mixed with Bituminous Coal.—During the war period large tonnages of anthracite culm and slush were shipped to industrial plants for mixing with soft coal. As the anthracite usually contained considerable refuse and mud and the mixing often was not well done, the best results were not obtained. However, a number of plants operated fairly satisfactorily with this mixture. The following results of tests on hand-fired four-drum Sterling boilers of 400 hp. rating using various mixtures of slush and soft coal were obtained:

Proportion of soft coal, per cent	10.7	18.9	26.2
Size of slush, on $\frac{3}{32}$ in. round mesh, per cent	4.8	9.0	7.3
on $\frac{3}{64}$ in. round mesh, per cent	41.8	47.4	48.4
through $\frac{3}{64}$ in. round mesh, per cent	58.2	52.6	51.6
Builder's rating developed, per cent	117.8	118.6	127.5
Equivalent evaporation per pound dry coal, pounds	6.95	7.57	7.95
Ash in dry coal, per cent	23.45	16.99	18.65
Fuel cost per 1000 lb. equivalent steam, cents	13.94	13.51	13.48

Pulverized Slush.—Burning of slush in a pulverized state has engaged the attention of a number of investigators and at least two of the important anthracite companies have made extended tests on a commercial scale. The most successful work has been done by the Susquehanna Collieries Co., which has been operating two plants—one at Lykens and the other near Minersville—for some time. The installation at Lykens has been quite successful, as shown by the results reported by J. R. Wyllie, Jr.[6]

	Average	Maximum
Moisture, per cent	0.53	0.78
Volatile matter, per cent	8.61	9.34
Fixed carbon, per cent	78.70	79.85
Ash, per cent	12.16	13.47
Heating value per pound, dry, B.t.u.	13,270.0	13,509.0
Pulverized through 100 mesh, per cent	95.27	97.00
Pulverized through 200 mesh, per cent	82.78	86.73
Builder's rating developed, per cent	153.8	228.6
Equivalent evaporation per pound of dry fuel, pounds	9.12	10.50
Overall efficiency, per cent	66.7	77.7

[6] *Jnl.* Engrs. Club of Philadelphia (Feb., 1921).

Mr. Wyllie points out that improvements made while the tests were being run added greatly to the efficiency and capacity obtained so that the maximum results shown are more nearly representative of the results they are now regularly getting.

Lykens slush is peculiarly adapted to burning pulverized because of its high volatile content, low ash, and friable nature. Most anthracite slush, however, carries one-half the volatile matter, is much harder, and, unless purified by concentration, carries over double the ash content. The slush at the Susquehanna's Lytle colliery, near Minersville, is a good example of the average anthracite slush, and here difficulty was at first experienced. After some experimentation all combustion and boiler-operating difficulties were overcome but the pulverizing problem proved less easy of solution.

Recent operation with a Hardinge mill indicates that the pulverization of anthracite slush has been put on a practical and economical basis. Good capacity has been obtained and power, supplies, and maintenance costs are well within the margin allowable. As an additional improvement in the preparation of the slush for pulverized use, Deister-Overstrom concentrating tables have been installed at Lytle colliery to remove the refuse and so reduce pulverizing costs.

It now can be stated that coal can be recovered from slush, cleaned, dried, pulverized, and delivered to the boiler room at a cost approximating the average selling price of barley coal.

It is estimated that about 12 per cent. of total anthracite production, or approximately 9,500,000 tons, is used yearly for mine fuel. It is probable that the average efficiency of the boilers in use is 50 per cent. With the use of pulverized coal, this efficiency can be raised to 70 per cent., and thus reduce the tonnage of steam coal required to 6,800,000 tons annually. Nearly two-thirds of this tonnage, or 4,000,000 tons, can be recovered from slush so that a total annual saving of 6,700,000 tons is possible. The coal saved would be mostly barley and rice sizes and would have an average value of about $1.75 per ton. making the money saving over $10,000,000 annually. With the return of normal industrial conditions and because of the better demand for rice and barley sizes that have been concentrated to a uniformly low ash content, little difficulty should be experienced in selling this extra tonnage of steam sizes.

Conclusions

1. Anthracite breaker slush contains annually about 9,500,000 tons of total solids of which 4,000,000 tons are recoverable with an ash content of 15 per cent.

2. Cost of recovering the usable coal in slush will average 15 to 20 c. per ton of dry product.

3. If the recoverable coal in slush is converted into domestic fuel, it will add 9 per cent. to present domestic shipments of anthracite.

4. If the recoverable coal in slush is used, pulverized, in mine boiler plants it will produce 59 per cent. of the power required to operate the mines and make available for other purposes about 5,600,000 tons annually of barley and rice coal now so used.

5. Complete prevention of pollution can be accomplished at a cost, except under unusual conditions, of from 1 to 2 c. per ton of breaker shipments, provided no value is placed on the recovered coal.

6. If the recoverable coal in slush is valued at not over 35 c. per ton, such value will pay the cost of recovery and also the cost of preventing stream pollution by the remaining slush solids that at present have no commercial value.

DISCUSSION

HENRY FLOOD, JR., New York, N. Y. (written discussion).—The Superpower Survey, recently conducted under the auspices of George Otis Smith of the U. S. Geological Survey, and under the immediate direction of W. S. Murray as its chairman, found two conditions that should eventually have much to do in aiding in the disposal of the smaller sizes of anthracite.

The first is the large diversity in the load characteristic in the anthracite region and in the industrial region along the Atlantic seaboard. This diversity permits large power plants located in the anthracite region, maximum load of which (being controlled by the mining industry) falls in the morning, to transmit a large portion of their plant capacity and energy into the seaboard industrial region at the time of the latter's maximum load, which falls usually about 5 o'clock in the afternoon; by that time, the load in the anthracite region is largely diminished. This results in a marked saving in investment in power plants and also in the production cost, due to the high load factor under which the large power plants in the mining region can be operated.

It is estimated that in 1919, the load along the seaboard that could have been economically supplied by the electric utility systems was as follows:

	Kilowatt-hours
Electric utilities	5,200,000,000
Manufacturing industry	1,900,000,000
Heavy traction railroads	1,100,000,000
Total	8,200,000,000

To this must be added the load within the anthracite region itself, which was estimated for 1919 to be:

	Kilowatt-hours
Electric utilities	740,000,000
Mines and industries	1,520,000,000
Heavy traction railroads	740,000,000
Total	3,000,000,000

In 1919, there were available as electric utility load, about 11,200,000,000 kw.-hr., which, it is estimated, will grow, by 1930, to about 18,000,000,000 kw.-hr., made up of 14,200,000,000 kw.-hr. for the electric utilities, mines, and industries, and 3,800,000,000 kw.-hr. for the heavy traction railroad electrification. This total energy required is over one-half of that for the entire superpower zone.

The second condition found by the Survey, pointing toward the establishment of large power plants in the anthracite regions, was the present costly method of supplying power to the anthracite mines, principally by inefficient isolated power plants that use for power purposes nearly 8 per cent. of the anthracite mined. These plants offer an excellent market for cheap power made in large steam-electric plants and at the same time, through their abandonment, would release a sufficient amount of the smaller sizes of coal for the larger plants to be operated.

The question of condensing water for large power plants is a most serious one, for unless water can be had in sufficient quantity, the efficiency of the plant is materially reduced through lowered vacuum and a large part of the benefits accuring to such plants are thus lost. In order to supply plants of 300,000 kw. capacity, a minimum flow of about 1200 cu. ft. per sec. is required. The Susquehanna River, in the vicinity of Wilkes-Barre and below, will support such plants. The Delaware River, from Belvidere down, will support, on its natural flow, plants of about 200,000 kw. capacity.

To put power plants into successful operation for the joint use of the anthracite region and the industrial centers along the seaboard, such plants must be constructed on a very large scale, or the cost of transmitting energy to the seaboard offsets the benefits gained. It should be possible to produce power in steam-electric plants in the anthracite region for 5.7 mills per kilowatt-hour at their switchboards, even with the high costs prevailing in the mid-year of 1919. Under like circumstances, power can be produced with bituminous coal at New York for about 7.4 mills, and at Philadelphia for about 7.3 mills. These costs are predicated upon No. 3 buckwheat at $2.75 per long ton delivered to the plant and Clearfield coal delivered to the New York plant at $5.22 per short ton and to the Philadelphia plants at $5.05 per short ton.

By 1930, the anthracite mining region should successfully support about 900,000 kilowatts in new steam-electric plant capacity in addition to about 110,000 kilowatts of the present more efficient central stations burning anthracite fuel, which should be retained and incorporated into the system.

These plants should generate about 6,400,000,000 kw.-hr. annually

of which 3,600,000,000 kw.-hr. would be required for the needs of the anthracite region and the remaining 2,800,000,000 kw.-hr. would be exported to the seaboard load centers. By 1930, on the basis of this amount of power exportation, the transmission of this power could be effected for about ½ mill per kw.-hr. generated, and the cost of power delivered at seaboard could be effected for about 6½ mills, which would effect a saving in the total cost of power at seaboard of between 11 and 12 per cent. of the cost of production.

By 1930, (should the recommendation, of the Superpower Survey be placed in execution) through the operation of about 1,000,000 kw. of plant capacity in the anthracite region, nearly 6,300,000 tons of the smaller sizes of coal would be required for these plants. According to Tryon, of the U. S. Geological Survey, there were 2,377,000 tons of buckwheat sold in 1919 and about 7,787,000 tons used in the isolated power plants at the mines, making a total of over 10,000,000 tons of the usable small sizes available in that year. Tryon estimates that by the complete electrification of the mines, releasing the coal now used in their plants, that 9,000,000 tons of steam sizes would be a conservative quantity to consider as available for some years to come. This is nearly 3,000,000 tons in excess of that required by the power plants, but this excess is not very much greater in amount than the present sales of No. 3 buckwheat. However, by the coöperation of the mine operators and power interests, it will be possible to make the quantity of excess buckwheat available for sale meet almost any market condition within wide limits through the proper operation of the power plants. Thus, by the simultaneous electrification of the mines and construction of 300,000-kw. power plants in the mining region, not only can the requirements of the mines be taken care of at a much lower cost than is possible today, but the smaller sizes of coal can be practically absorbed and, at the same time, the industrial centers near the Atlantic seaboard can be supplied with the surplus power at a lower cost than that for which they can generate it themselves. The results stated have all been based on the use of No. 3 buckwheat coal.

With our present knowledge of combustion, it is entirely feasible to design large power plants for one grade of fuel that later, at a very small expense, may be changed to use an inferior grade and as soon as efficient methods are developed for using the very small sizes of anthracite, production costs even less than those here given can probably be obtained.

There is considerable research under way in an effort to make these very small sizes of coal usable, and it is important that these investigations be continued with increasing activity and that a comprehensive plan for carrying out such investigation be put into effect.

There are certain underlying conditions that must be met if the rec-

ommendations that will be made in the Superpower Report are placed in execution:

1. It must be possible for the power-plant owners to contract for a stable supply of small sized coal for a period equal at least to the reasonable life of the plants; that is to say, from 30 to 40 years. This has been successfully accomplished in a number of locations in the bituminous fields, although conditions there differ materially from those in the anthracite fields.

2. The contract price for coal over this period must be such that power can always be delivered to the seaboard with added transmission cost at a lower price than the cost of producing there with bituminous coal. The steam sizes of anthracite have always had to compete with bituminous coal at tidewater, as a result of which, certain relations in price have been fairly well established. For instance, from 1911 to 1917, the selling price per long ton for No. 3 buckwheat at New York varied between 53 and 56 per cent. of the price per short ton for Clearfield coal at the same destination point. However, in 1918, this rose to 67 per cent., but dropped back to 66 per cent. in 1919. These increases were undoubtedly due to the governmental regulation of bituminous coal prices, combined with the scarcity in the supply of bituminous coal. Just what the ratio is today under present chaotic market conditions it is hard to say. However, the point I wish to establish is that such contracts would not have to be based upon a fixed price, but upon some competitive price compared to that of bituminous coal at a definite delivery point. The gathering charge for transporting the coal from the mines to the power plants must be kept low enough so that the total cost of small-sized anthracite delivered to the plant will maintain its competitive position with bituminous coal delivered to the seaboard plants.

Most of the existing power plants supplying the anthracite mines are notoriously inefficient. In the past, this has been of little consequence, as the unmarketable sizes of coal had to be disposed of in some manner. The construction of large power plants should enable the power companies to offer attractive power contracts to the mines, which would reduce the power item of mining costs. In turn, the miner could contract for the disposal of the small sized output to the power company, and taking these two forms of contract into consideration, arrangements could be made that should prove mutually profitable and satisfactory to both sets of interests.

It is rather difficult to estimate just what these savings will be in the anthracite district, as distinguished from the balance of the Superpower zone, but based on the district's power requirements and on mid-year 1919 costs, they should be about $60,000,000 yearly.

Anthracoal: A New Domestic and Metallurgical Fuel

BY DONALD MARKLE, HAZELTON, PA.

(Wilkes-Barre Meeting, September, 1921)

ANTHRACOAL is a mixture of small particles of anthracite coal and a matrix of practically pure carbon, formed from the distillation of coal-tar pitch or other suitable bitumen. It is a hard, dense, homogeneous mass, with a silvery luster and in color varies from silvery to grayish black. When pushed from the oven, it develops only partly the fingerlike structure of coke; but, unlike coke, it has a tendency to remain in blocky masses. When struck with a hammer or passed through crushing rolls, it breaks with an irregular fracture, similar to anthracite, but with very little fines. Due to its density, anthracoal is harder, tougher, and stronger than coke.

The results of a test made by the blast-furnace department, of the Bethlehem Steel Co., on two barrels of anthracoal are given in Table 1.

TABLE I.—*Results of Shatter Tests on Anthracoal and Coke*

Test	Anthracoal		Good Blast-furnace Coke
	First Barrel	Second Barrel	
Moisture, per cent.	0.77	0.76	Under 5, variation not over 3 points
Sieve test			
Through 2-in. screen, per cent...	7.71	7.29	Under 40
Through 1-in. screen, per cent. .	1.37	1.56	
Through ½-in. screen, per cent.	0 93	1.08	Under 8
Over 2-in. screen, per cent........	92.29	92.71	Over 60
Shatter test			
Through 2-in. screen, per cent. . . .	10.06	12.40	Under 16
Through 1-in. screen, per cent...	2.66	2.06	
Through ½-in. screen, per cent. .	1.80	1.26	
Over 2-in. screen, per cent..... . .			
Hardness number....	86.40	86.10	Over 81
Analysis			
Ash, per cent.	16.64	16.48	Under 11
Sulfur, per cent....................	1.17	1.10	Under 0.95
Volatile matter, per cent...........	0.77	0.27	
Fixed carbon, per cent.............	82.59	83.25	Over 87

Both the shatter tests and the hardness number indicate that anthracoal is an extremely tough hard material that is eminently fitted to resist rough handling, crushing, and abrasion.

Anthracoal can be made in a coke oven upon the same large scale as bituminous coke and can be produced with little greater expense; therefore it should prove a tremendous factor in utilizing the anthracite culm now going to waste. Its commercial development is the outcome of experiments, made in 1914, in the chemistry laboratory at Lehigh University. In the development of the process, the services of W. H. Blauvelt, were obtained and through him the Semet-Solvay Co. became interested in the experiments.

Inasmuch as the success of anthracoal will depend not only on its manufacture on a large scale but also on its salability, the tests at Syracuse were made to duplicate, as far as possible, actual commercial conditions. Although the apparatus was not designed for this purpose and the ovens were rather obsolete, sufficient anthracoal was made on a large scale to indicate the following:

1. The process is practicable and the only difficulties encountered are of a mechanical nature that can be remedied.

2. Anthracoal has demonstrated, by tests and actual use, its excellent qualities as a domestic fuel.

3. Several carloads of anthracoal were shipped to different retail dealers, who reported that the customers were satisfied with the product and had no difficulty in burning it. Also, anthracoal commanded the same price as the best anthracite, and the customers to whom it was sold asked for more.

Analyses of Anthracoal

Table 2.—*Analysis of Anthracoal in Coke Ovens at Syracuse*

Oven No.	Apparent Specific Gravity	True Specific Gravity	Porosity, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent	Ash, Per Cent.	Sulfur, Per Cent
7	1.046	1.717	39.2	1.35	79.85	18.18	1.11
10	1.124	1.896	40.8	3.23	76.13	20.64	1.15
13	1.015	1.649	38.5	2.65	80.00	17.35	1.15
22	1.004	1.636	38.7	2.90	77.94	19 16	1.16
38	1.044	1.696	38.5	2.22	79.23	18.55	1.06

Table 2 shows the analyses of anthracoal made in different ovens from culm from the Loree breaker of the Hudson Coal Co., and are typical of the product. The apparent specific gravity is about 1.05, while that of coke is about 0.84, showing that anthracoal is heavier and denser than coke. The porosity is about 39.1 per cent. while that of coke is between

50 and 55 per cent. This accounts, in a large measure, for the greater length of time that anthracoal burns, as compared to coke, unless the latter is carefully watched and burned on a bed of ashes. The fact that the domestic consumer does not know how to prevent coke from burning so quickly has prevented its competition with anthracite as a domestic fuel.

To demonstrate that the fuel may be greatly improved by reducing the ash in the culm, some culm received from the Lehigh Coal & Navigation Co., which ran 35 per cent. ash, was reduced to 9 per cent. ash by the use of a Wilfley table; the anthracoal produced from it gave the following analysis:

Analysis of Anthracoal Made from L. C. & N. Cleaned Culm

Moisture, per cent.	0.20	Sulfur, per cent.	0.52
Volatile matter, per cent.	1.41	Apparent specific gravity	1.101
Fixed carbon, per cent.	89.99	True specific gravity	1.8
Ash, per cent.	8.4	Cellular space, per cent.	38.85
		B.t.u.'s per lb.	13,334
	100.00		

By cleaning the culm a product results that is superior to anthracite and more than meets every requirement of the best byproduct coke. The sulfur is reduced from 1.15 per cent. to 0.52 per cent., which is materially lower than the blast-furnace limit of 0.95 per cent. and the ash is considerably lowered, thus meeting the ash requirement of byproduct coke, under 11 per cent.

Tables such as the Diester-Overstrom should be used to reduce the ash and sulfur in the culm, there is no reason why the ash cannot be commercially lowered to 10 or 12 per cent. This would produce a superior grade of anthracoal, which could be sold at a premium, as the best grade of nut anthracite today rarely runs lower than 17 and 18 per cent. ash, and often is as high as 22 per cent. Small particles of coal carefully picked from the culm contained from 2 to 3 per cent. ash, hence the inherent ash in the culm is about 2.5 per cent., showing that the limit to which the separation may be carried is about 3 per cent. ash. However, while 3 per cent. is the theoretical limit to which the separation may be carried, 7 or 9 per cent. ash is a conservative figure and by careful preparation may be realized.

Process of Manufacture

The process of manufacturing anthracoal is comparatively simple; in fact, it is practically identical with manufacturing coke, except that a binder must be added to the culm and the mixture well ground and mixed before coking. The culm is first dried and then placed in a hopper whence it runs upon a proportioning belt. Another proportioning belt, run at the

same speed, carries the pitch. By the use of weirs and the regulation of the openings, the amount of pitch and culm passing on to the belts are regulated to any desired proportion. The proportion, by weight, used successfully at Syracuse was 17 per cent. pitch to 83 per cent. culm. Both belts dump into the same bin, from which the mixture is run into a grinder where the culm and pitch are well ground and mixed but not pulverized. The mixture is then elevated from the grinding mill to the charging bins above the ovens; from this point on the process is identical with the coking of bituminous coal.

Depending on the market conditions, anthracoal is sized to egg, stove, nut, pea, and breeze. Coke for domestic trade is sized and crushed

FIG. 1.—ANTHRACOAL BEING PUSHED FROM OVEN.

in the same manner. For blast-furnace purposes, however, run-of-oven coke is used. This would be true also of anthracoal used for metallurgical purposes. The breeze produced in crushing anthracoal is less than that produced in crushing anthracite or coke. In a properly designed crushing and sizing plant, where anthracoal will be made in the same sizes as anthracite, the breeze should not exceed 5 per cent.

The coking time of anthracoal, compared with soft coal, is a most important factor, as any shortening in time increases the yield and decreases the cost per ton of manufacture. In the experiments at Syracuse, ovens of anthracoal were pushed in 17 hr. against 19 hr. for bituminous coal under the same conditions. It is safe to assume that in a modern

regenerative oven, where the heats are much greater, the coking time of anthracoal will be 16 hr. against 18 hr. for bituminous coal.

The percentage of pitch used varies somewhat with the pitch used, the character of the culm, and the method of mixing and grinding the material. In the tests at Syracuse, two grades of pitch were used, one with a melting point of 265° F., and one 280° F.; both worked well. The amount of pitch varied between 14.8 and 25 per cent., due to the method of proportioning. Ovens that contained 15 per cent. pitch and 85 per cent. culm, by weight, gave an excellent anthracoal and pushed readily. Ovens that contained 25 per cent. pitch and 75 per cent. culm also gave an excellent anthracoal but more of a thick carbon scale was noted in the ovens than when the mixture contained less pitch; also, the anthracoal was more porous than that made from the 15 and 17 per cent. mixtures. After many tests, both in the coke ovens and in the laboratory, it was found that between 16 and 17 per cent. produced the best anthracoal with minimum fines and scale. Using below 16 per cent. pitch, the anthracoal showed signs of attrition and not holding together, while above 17 per cent. the surplus pitch flowed to the sides of the retort and produced a carbon scale that made a great deal of fines when pushed on to the dock.

The pitch used in the experiments, obtained from the Barrett company, was the product obtained in the byproduct coke industry after all the valuable constituents of the coal tar had been removed. It may be obtained in either lump or flake form. For these experiments, the flake form was used, as the flakes are little larger than the culm and hence could be ground with the culm without previous preparation. Its analysis is as follows:

Flake Pitch, No. 1		Flake Pitch, No. 2	
Melting point, degrees F.	265	Melting point, degrees F.	280
Moisture, per cent.	0 11	Fixed carbon, per cent.	46 64
Fixed carbon, per cent.	44.55	Volatile matter, per cent.	53.08
Volatile matter, per cent.	54.39	Ash, per cent.	0.28
Ash, per cent.	0.95		

A few analyses of culm used are here given:

Volatile matter, per cent.	6.96	6.51	6.52	7.42
Fixed carbon, per cent.	73.49	74.22	77.85	69.6
Ash, per cent.	19.55	19.27	15.63	22 98
	100.00	100.00	100.00	100.00
Sulfur, per cent.			1.26	1.96

The average ash content of the culm used from the Loree breaker ran 18.96 per cent. the volatile matter ranged from 7.42 to 6.52 per cent. The sulfur varied from 1.26 to 2.5 per cent. All the culm used passed through a $\frac{3}{64}$-in. (1.19 mm.) round-mesh screen and was taken

from the fresh-mined side of Loree breaker, hence it was lower in ash than the regular run of culm, which is about 35 to 40 per cent. ash.

Fineness of Pitch-culm Mixture Before Charging into Ovens

Many tests were made, both in the laboratory and in the ovens on a large scale, to learn the proper fineness of the pitch-culm mixture that would produce the best anthracoal. The first ovens pushed were made from a mixture of culm and pitch that had not been ground. That is, the culm was in the state in which it was received from the mines. The anthracoal produced was hard, coarse, and had a bright silvery luster. Each particle of culm could be readily seen; the anthracoal, however, had a decided tendency to scale and particles of culm would easily rub off, showing that if this material were to be shipped a great deal of fines would result, which would prove most unsatisfactory to both dealer and consumer. By grinding the pitch and culm together, so as to reduce the larger particles of culm, a dense, homogeneous, strong anthracoal was made, which gave no signs of attrition and withstood the roughest handling. Besides, the grinding gave a much better mixing of the materials than could be accomplished otherwise and resulted in a very uniform product. A screen test of the culm as received from the mines is as follows:

Screen Test on Culm Before Grinding

	Per Cent.
On 20 mesh	7.57
On 40 mesh	48.15
On 60 mesh	28.40
On 80 mesh	6.15
On 100 mesh	3.60
On 200 mesh	4.45
Through 200 mesh	1.58

Screen Test on Pitch-culm Mix after Grinding

	Per Cent.	Per Cent.	Per Cent.
On 20 mesh	2.73	2.80	3.20
On 40 mesh	31.11	23.20	24.40
On 60 mesh	28.76	26.93	27.06
On 80 mesh	7.32	10.00	9.65
On 100 mesh	3.71	6.90	5.61
On 200 mesh	22.04	15.10	19.23
Through 200 mesh	4.33	15.07	10.85

A comparison of these tests shows that little grinding is required to get the mixture to the proper fineness for charging. The fineness necessary for pulverized fuel is not desired.

Weights and Yields

The weight of 1 cu. ft. (0.028 cu.m.) of bituminous coal, prior to being charged into the oven, was 43.5 lb. (19.7 kg.), the weight of 1 cu. ft. of the pitch-culm mixture containing 83 per cent. culm and 17 per cent. pitch prior to charging was 59 lb. (26.7 kg.), or a difference of 15.5 lb. (7.02 kg.). That is, the same volume of pitch-culm mixture weighs 35 per cent. more than the same volume of bituminous coal; therefore, a little over 7 tons

of pitch-culm mixture could be charged into the same oven as 5.3 tons of bituminous coal.

In modern coke-oven practice, the yield of coke per ton of bituminous coal charged is about 70 to 72 per cent., depending on the quality of the coal. In practice, all the volatile matter is not driven off, a certain small percentage being precipitated as free carbon, which passes into the coke, thereby increasing the weight of the coke about 2 or 3 per cent.; thus, a 32 per cent. volatile coal, instead of giving a yield of 68 per cent. coke, would give 70 or 71 per cent.

When using the pitch-culm mixture similar results are obtained. The mixture containing 17 per cent. pitch and 83 per cent. culm has 15 per cent. volatile matter; therefore the theoretical yield of anthracoal should be 85 per cent. But as some of the volatile matter is precipitated as carbon the actual yield is 87 per cent. An oven was charged with a mixture of 23 per cent. pitch and 77 per cent. culm, which contained 18 per cent. volatile matter; after the oven was coked and pushed, the anthracoal was carefully weighed and showed a yield of 84 per cent.

Another mixture of 17 per cent. pitch and 83 per cent. culm was prepared, and weighed carefully in the laboratory, then charged into a bomb and all the volatile matter driven off; the yield of anthracoal was carefully weighed and was found to be 87 per cent. exactly; so that the figures have been checked by both actual tests on a large scale and by check runs in the laboratory.

The fact that more of the pitch-culm mixture than bituminous coal can be charged into an oven is most important. For instance, the coking time of bituminous coal is 18 hr., therefore if an oven will hold 16 tons, 21.28 tons will be charged in 24 hr. If the coal contains 30 per cent. volatile matter, the yield in coke will be 72 per cent., or 15.32 tons. As the same oven can be charged with 21.12 tons of the pitch-culm mixture and as the coking time is only 16 hr., in 24 hr. 31.68 tons will be charged. As the mixture contains 15 per cent. volatile matter, the yield will be 87 per cent. of 31.68 tons, or 27.56 tons, a difference of 12.24 tons in favor of anthracoal.

Byproducts Obtained

Naturally, the byproducts resulting from the culm-pitch mixture used in making anthracoal are not present in as great quantities as in the manufacture of coke from soft coal. Also, a smaller amount of gas is given off by the mixture when heated. Bituminous coking coal contains from 28 to 35 per cent. volatile matter, while a mixture of 17 per cent. pitch and 83 per cent. culm contains only 13.73 per cent., showing that the bituminous coal will give off more than twice as much gas and byproducts as the anthracoal mixture.

From the byproduct results obtained in the various tests made, the

following average data as to yields per ton of anthracoal charged have been obtained:

Ammonium sulfate, pounds per ton	9.9
Light oil, gallons per ton	0.6
Tar (above 170° C.), gallons per ton	5.07
Gas (from oven test), cubic feet per ton	4080
Gas (from bomb test), cubic feet per ton	5910
Heating value of gas (determined), B.t.u. per cubic foot	330
Heating value of gas (calculated), B.t.u. per cubic foot	336
Anthracoal yield, per cent.	87.0

Analysis of Gas

	Per Cent.		Per Cent
Carbon dioxide (CO_2)	1.0	Carbon monoxide (CO)	3.1
Benzene (C_6H_6)	0.1	Methane (CH_4)	9.0
Ethylene (C_2H_4)	0.1	Hydrogen (H_2)	73.4
Oxygen (O_2)	1.0	Nitrogen (N_2)	12.3

Analysis of Material Used in Bomb Tests

	Washed Culm, Per Cent.	Pitch, Per Cent.	Mixture 17 Per Cent. Pitch–83 Per Cent. Culm, Per Cent.
Volatile matter	6.56	56.2	13.73
Ash	8.05	0.25	6.86

Analysis of Material Used in Coke Oven

	Per Cent.
Volatile matter	15.03
Ash	16.20

These tests and data show that the light oil is not present in sufficient quantities to make it worth while recovering; therefore the only possible byproducts are the gas, ammonium sulfate and the tar. Whether or not the installation of the apparatus necessary to recover the ammonium sulfate and the tar is worth while is a matter for future study. At the present time, the writer would say that no money should be spent on a byproduct recovery plant for making anthracoal. He would use the non-recovery type of oven, which eliminates the byproduct recovery apparatus and burns the gas evolved from the charge directly in the flues. Besides being less expensive, this type of oven will generate greater heat from the gas by being burned directly in the flues. This oven should not only prove self-sustaining in heat value but should be able to coke the anthracoal in less time than the recovery type of oven.

Gas

The gas results are based on two series of tests, one of which was made in the byproduct apparatus on a full oven charge and the other by means of the standard Semet-Solvay bomb test method. The results obtained in

the bomb test indicate a greater yield of gas than the results from the oven test, due to the following:

1. The bomb tests were made on low-ash (8.05 per cent.) slush prepared on a concentrating table to approximate the slush that is to be used on a commercial scale, while the oven tests were made on the high-ash (18 per cent.) slush. This fact accounts for 11.2 per cent. more gas than the oven tests show, which may be expected on a commercial scale.

2. There is some leakage in the gas apparatus at the ovens, which also makes the gas yield low on the oven test, while the bomb test allows no leakage and records all gas.

3. The bomb test records all the gas obtained in the charge, while the oven test does not, as some of the gas is left in the charge as unvolatilized matter.

4. The oven temperature on the oven test were below normal.

Taking the above facts into consideration and calculating the oven-test results to low-ash slush, gives the following summary of low gas yields and heating values.

	Gas per Ton, Cubic Feet	Heating Value, B.t.u. per Cu. Ft.	Total Heating Value, B.t.u. per Ton
Bomb test	5,910	336	1,985,760
Oven test	4,570	330	1,508,100
Average	5,240	333	1,746,930

These results indicate that with a non-recovery type oven there would be sufficient gas for operation but that with a byproduct recovery oven the self-sustaining operation would be doubtful.

The basis of this conclusion is as follows:

Average gas yield from bituminous coking operation, cubic feet	10,000
Average heating value of gas, B.t.u. per cubic foot	500
Total heating value per ton of bituminous coal, B.t.u.	5,000,000
Average gas used for coking, per cent.	50
Heating value in gas to coke 1 ton bituminous, B.t.u.	2,500,000

21 tons of anthracoal charge coked in 16 hr. }
16 tons of bituminous coal coked in 18 hr. } by same amount of heat

31.5 tons of anthracoal and }
21.6 tons of bituminous coal } coked in 24 hr. by same amount of heat.

1.46 tons of anthracoal or }
1 ton bituminous coal } coked by 2,500,000 B.t.u.

1 ton of anthracoal charge would require $\frac{2,500,000}{1.46} = 1,712,000$ B.t.u.

The bomb test results indicate an excess of gas while the oven test results indicate a deficiency; the average shows a slight excess. With a non-recovery oven, the margin would be considerably greater, as all the sensible heat in the gases, besides the byproducts, would go back into the heating of the ovens.

Comparative Eight-day Burning Test in Kitchen Range

A direct comparison between chestnut size of anthracoal and chestnut size of anthracite coal was made by an eight-day test of each in a kitchen range. The range performed the same service in each test, cooking three meals a day, heating water in the boiler, and, in fact, doing exactly the same work in each case. Observations were made as to amount of fuel fired each day, the amount of ashes produced, and the position of the draft during each run. The fire was started with wood at the beginning of each test and allowed to burn completely out at the end of the eight days. The fire was dampened at night and between meals with equal ease with the anthracoal and anthracite.

The total fuel consumed in each eight-day test was 288.5 lb. (130.8 kg.) of anthracoal and 346.52 lb. (157 kg.) of anthracite, or 20.1 per cent. more anthracite than anthracoal. As exactly the same service was obtained from each fuel, the results show that in this range the chestnut anthracite was 20 per cent. less efficient than the anthracoal of the same size.

Summary of Tests

	Chestnut Anthracoal	Chestnut Anthracite
Fuel fired, pounds	288.50	346.52
Ashes removed, pounds	59.5 (20.6 per cent.)	61.9 (17.9 per cent.)
Number of days burned	8	8
Number of hours burned	192	192
Fuel burned, pounds per hour	1.50	1.80
Relative efficiencies, anthracoal = 100 per cent	100	80

These tests were made on commercial anthracite prepared by the Hudson Coal Co. at Marvine colliery and should be representative of the anthracite placed on the market at the present time.

Conclusion

The evidence and data submitted thus far appear to point to a plant of regenerative non-recovery ovens; that is, a plant where the gas is brought directly from the charge into the flues and regenerative apparatus of the oven instead of passing through cooling coils and other apparatus for abstracting the byproducts before it is burned.

The reasons for doing this are as follows:

1. If the gas is cooled first the sensible heat will be lost.
2. Removal of the byproducts will lessen the heating value of the gas.
3. The byproducts in the anthracoal gas are about one-third to one-half those in the bituminous gas, so that it is doubtful, with the uncertain and changing market, whether the recovery of the byproducts would be worth the expense of the byproduct apparatus which represents one-half the cost of the ovens.

4. A non-recovery oven of the regenerative type will burn the gas direct, have the benefit of the sensible heat in the gas as it comes off the charge, and also have the additional heat value of the byproducts which are not removed.

5. The ovens should prove to be self-sustaining and have ample gas for an exceptionally hot retort and quick coking in the non-recovery type.

6. A non-recovery type oven operating with anthracoal should produce 79 per cent. more anthracoal in 24 hr. than the same type of recovery oven. As the cost of manufacture in each case is almost the same, with perhaps a little greater expense involved in the preparation of the anthracoal mix, the 79 per cent. greater production of anthracoal than coke in 24 hr. more than offsets the loss of the byproducts.

7. A non-recovery type oven should prove more adaptable to the anthracite region. Its cost will be less to construct than the recovery type, and it will not require a skilled staff of chemists and workmen.

8. No market need be developed or additional sales force required, as in the byproduct game, but rather the maximum amount of fuel can be produced, which can be handled in conjunction with the sale of anthracite coal.

Naturally, the success of anthracoal will not be proved until a plant has been erected and the ovens operated on a commercial scale for a period of time. As a non-recovery regenerative oven has been rarely used in this country, it will embody some changes in design from the regular regenerative type. However, this type of oven has been used with success in Germany and has not presented any difficulties, as far as can be learned.

DISCUSSION

William Griffith,* Scranton, Pa.—Has the author any analysis of the ingredients used in these comparative tests? Has he an analysis of the chestnut coal used?

Donald Markle.—For the eight-day tests?

William Griffith.—Yes, because otherwise the tests are rather indefinite and misleading.

Donald Markle.—As I remember the figures, the ash in the anthracite was a little less than that in the anthracoal. The chestnut coal was from the Marvine colliery and, I believe, ran about 5 per cent. volatile, and 16 per cent. ash; the anthracoal ran about 18 per cent. ash.

Edwin Ludlow,* New York, N. Y. (written discussion).—This is one of the possible solutions of a problem that the whole anthracite region

* Consulting Engineer.

should face; the utilization of the fine slush now going to waste by manufacturing it into the equivalent of a domestic fuel.

The anthracite operators have reached the point of maximum output and a continually increasing demand, which requires the utilization of everything brought out of the mines in order to meet the increasing cost of operating and the demands for anthracite fuel. No accurate figures are available as to the amount of slush, or coal finer than the commercial buckwheats now sold, that is thrown away, but a conservative estimate would be 5 per cent. This would amount to practically 4,000,000 tons a year, containing from 10 to 40 per cent. of ash, the utilization of which would go far toward meeting the continually increasing demand for domestic fuel.

Mr. Markle's tests have been with pitch and an ordinary culm; but the experimental results, though not on a commercial scale, show the success of the process from a dollars-and-cents point of view. His proposition to use non-recovery ovens is feasible, and they should produce a surplus of gas above what is required for the heating of the ovens which could be used directly under boilers for the generation of steam. This has been done at a great many plants, but it has the objection that the byproducts are lost.

I would suggest that Mr. Markle's experiments be carried on with the amalgam produced by the Trent process. This process consists of the thorough churning of a mixture of 70 per cent. of coal with 30 per cent. of oil or gas-house tar, with the addition of water; after the churning the coal and oil form an amalgam while the ash remains with the water. In this way the amalgam contains only from 2 to 5 per cent. of ash, has a B.t.u. value of about 15,000, and eliminates all necessity for any other form of purification of the slush used. The coal is ground to 200 mesh, and in the agitating process each particle of coal becomes coated with a thin film of the oil; this has the same effect on the distillation as is accomplished by the cracking process under pressure. The results obtained by using very heavy asphalt-base crude oils were extremely satisfactory.

Quite elaborate tests have been made in the laboratories of the Bureau of Mines at both Pittsburg, Pa., and Berkeley, Calif., and it has been found that the low-temperature distillation of this amalgam gives unusually high yields of the valuable byproducts, especially gasoline and kerosene. Some of the recent tests made at Berkeley show 7½ more gallons of gasoline and 11½ more gallons of kerosene to the ton than was obtained by the separate distillation of the coal and oil. One of the peculiarities of this process is that it works with almost any kind of oil, from kerosene through to the heavier grades and also with gas-house tar. It is possible, therefore, to buy the cheapest grade of oil on the market, for the largest distillation returns are obtained from extremely cheap crude asphalt-base oil.

All the experiments that have been made with this process have been

laboratory tests, which have been written up by the Bureau of Mines. The tests that are now required are the commercial ones, necessitating special machinery and special distillation plants to obtain the best results.

One experiment in which I was greatly interested was the molding of the amalgam into briquettes, and putting those through distillation. The resultant briquette, when broken, showed that it was hard coke with a bright metallic luster; the briquette retained its form and could be used as a domestic fuel. The high recoveries in gasoline and kerosene, as well as the gas that can be obtained from this distillation, should make the process, when fully developed, an exceedingly profitable way of disposing of what is now a waste product. This matter might well repay research investigation, which might be conducted by the anthracite industry through its bureau of information, as the whole industry would be interested in finding a profitable method of changing slush into a commercial product.

Edwin M. Chance,* Philadelphia, Pa.—Does Mr. Ludlow think it is possible to produce the Trent amalgam from anthracite?

Edwin Ludlow.—I have seen it done. In the experiment mentioned, 5 lb. of finely ground anthracite was mixed with the same amount of water and then the oil was stirred in. At first, the oil made a slush, then as the quantity of oil was increased little particles formed, like butter as it comes in the churn; the final result was like caviar. After being worked, the material forms into a mass that can be washed and dewatered. It still carries 10 per cent. of water, which can be removed by pressure, but no amount of pressure will remove the oil. It seems to be a chemical combination that makes it an almost unique product.

Edwin M. Chance.—What intrinsic difference is there in the end product, that distinguishes anthracoal from carbocoal? Of course, it is clear that the two methods are basically different and that the raw materials are different, but I would like to know just what distinguishes anthracoal as such from carbocoal?

Donald Markle.—I cannot answer; I am not familiar with carbocoal. I understand that it is made by heating bituminous coal in a retort which takes off the volatile matter and volatile gases; the carbonaceous residue is then pushed from the oven, crushed, and mixed with tar obtained from the gases in the first process, formed into briquettes, which are then baked in a second retort, making a very hard briquette or coke-like fuel. Although I have not seen any carbocoal, I would say that there would be the same difference between it and anthracoal as there is between coke and anthracoal; that is, anthracoal is much denser than coke and as it is made up of small particles of anthracite held

* Engng.-Mgr. Day & Zimmerman.

closely together in a matrix of pure carbon, it loses none of its anthracite characteristics.

Edwin M. Chance.—What specific gravity has finished anthracoal?

Donald Markle.—The apparent specific gravity of anthracoal is 1.1, that of coke is 0.84.

Charles Dorrance, Scranton, Pa.—Carbocoal cannot be made from anthracite; coal made by the Smith process can be made only with bituminous coal.

E. W. Parker, Philadelphia, Pa.—In the manufacture of anthracoal, is the oven self-sustaining; that is, do the gases generated reheat the oven?

Donald Markle.—This is one point about which we are not absolutely sure. As far as we can estimate and as far as our experience has shown, in a non-regenerative type of oven that we propose to build, the oven will be self-sustaining; that is, enough gas will be obtained from the charge to heat the ovens. But there is no way of ascertaining that fact until a non-recovery regenerative oven is built.

George H. Ashley, Harrisburg, Pa.—One state geologist brings up the question as to the financial possibility of this process converting all the steam sizes of anthracite into bituminous coal. I am wondering if the difference in the selling price of household coal and anthracite steam coal will not cover the costs of manufacture, as not all the steam sizes of anthracite can be converted into anthracoal.

Donald Markle.— I cannot answer that question until a commercial plant is in operation, and the condition of the market tells us how far we can carry the process commercially on the larger steam sizes of anthracite.

Charles A. Burdick, New York, N. Y.—Is the use of briquettes increasing rapidly, or was it just the war necessity that caused the increased use in 1917 and 1918?

Donald Markle.—I am not a briquette man, so I cannot answer that question very well, but I know the briquetting industry was in existence long before the present war.

William Griffith.—If this process is introduced commercially, will there be sufficient coal tar to keep it going? Will not the amount of coal tar available be exhausted pretty soon? There is plenty of slush and culm, but will there be much coal tar if this product is made in large quantities?

Donald Markle.—We believe a large supply of this pitch is available, for it depends on the byproduct coke industry.

Edwin Ludlow.—Has a low-grade asphalt-base oil been tried instead of pitch?

Donald Markle.—Yes, but it did not work; it has worked with asphalt though.

George S. Rice, Washington, D. C.—In the manufacture of the anthracoal, must the particles be quite small, and if some of the larger sizes are used will they require fine grinding?

Donald Markle.—Anthracoal can be made from the slush as it comes from the breakers; but the coarser particles of slush can be seen in the anthracoal. If two pieces of anthracoal are rubbed together, the coarser particles will rub off. If a consumer placed a ton of that anthracoal in his cellar, he would have a great deal of attrition in his coal bin, which is objectionable. That is the reason we do not pulverize, we merely grind the slush and pitch together, so as to get rid of all the +10-mesh material; that is, we want to get practically 90 per cent. through a 10-mesh screen. The paper contains two tests on the fineness of the slush before and after grinding.

Power Installation at Coverdale Mine

By Charles M. Means, Pittsburgh, Pa.

(Wilkes-Barre Meeting, September, 1921)

A thoroughly modern coal-handling system has been installed at the Pittsburgh Terminal R. R. & Coal Co.'s new No. 8 shaft, or Coverdale mine, about 11 mi. (17.7 km.) from Pittsburgh on a spur of the Montour R. R. The Pittsburgh seam, which is the one worked, is reached at a depth of 342 ft. (104 m.).

Power is delivered to the mine at 22,000 volts over a transmission line of the bow-arrow type of construction. A private telephone line connecting the various mines and the main office in Pittsburgh is carried on the same poles. The high-tension line is brought to an outdoor, step-down, transformer substation, where three 833-kv.-a., single-phase, 60-cycle, oil-insulated, self-cooled transformers reduce the potential to 2300 volts. The transformers are protected by electrolytic, aluminum-cell lightning arresters, horn gaps, and choke coils. High and low-tension busses, together with fuses, choke coils and air-break horn-gap switches with their remote-control operating mechanism are mounted on a substantial structural-steel tower 25 ft. (7.6 m.) in height.

Mine Substation

The transformer substation is adjacent to a pressed-brick building that houses direct-current substation equipment together with switchboards, etc., for the control and distribution of power. This building also contains the hoist for the men and supplies.

Main Switchboard

The 2300-volt circuit is carried underground in a conduit from the transformer substation to an oil switch in the substation and supply hoist building. This is the main switch controlling the 2300-volt busbars mounted back to a twelve-panel black-slate switchboard. From the switchboard, 2300-volt circuits are carried to transformers feeding the street and house lighting systems and to other transformers furnishing power for the elevator and lighting circuits in the store building. Other 2300-volt circuits run from this switchboard to the main hoist building, fan house and auxiliary or man-and-supply hoist. High-tension wiring,

including current transformers, connections to oil switches, etc. at the back of the switchboard are painted vermillion, while the low-tension connections and wiring are painted black.

A part of the remote-control mechanisms for the oil switches back of the main switchboard are mounted on a sub-floor. To make this mechanism readily accessible for the purpose of inspection or adjustment, removable floor slabs of reinforced concrete were made to fit in behind the switchboard flush with the main floor.

Motor Generators

Direct current for the mine circuits is supplied by two motor-generator sets, which are arranged to operate singly or in parallel. Each set consists of a 433-hp., 2200-volt, three-phase, 60-cycle, synchronous motor, direct connected to a 300-kw., 275-volt, compound-wound, direct-current generator.

Mine Feeder Circuits

The main switchboard has two panels for the feeder circuits to the mine. One of these panels controls the underground circuit for coal-cutting machines and the other the circuit that will feed the trolley system, when the development of the mine makes it necessary to supplant mules and a storage-battery locomotive with motor haulage.

Two 500,000 cir. mil cables with 1000-volt, varnished, cambric insulation were used on each of the positive and negative sides of both the trolley and machine feeder circuits. These circuits are carried down the air shaft in rigid iron conduits that are clamped to the concrete shaft lining. All eight cables are suspended from a wooden spool near the top of the shaft collar. The negative lines are connected to a permanent ground underneath the substation floor.

Auxiliary Hoist

The hoisting equipment at the air shaft consists of a Vulcan hoist, with cylindro-conical drum, with herringbone gear drive from a 250-hp., 2200-volt, induction motor with wound rotor. The control equipment is so designed that the motor cannot be accelerated beyond a fixed rate. The hoisting engineer may throw his master switch or controller lever into the full on-position, but accelerating relays will allow magnetic switches controlling the resistance in the secondary circuit to close only at a predetermined rate. In addition, the hoist is fully protected by automatic safety devices to prevent overwinding or overspeeding. Limit switches are placed in the headframe at the height beyond which it is unsafe to hoist the cage; these switches give a second protection against overwinding. A device on the hoist framework shows when the links

of the brake rigging require tightening. All of these safety features are so connected with the control circuit of the hoist that should any one of them operate, power will be cut off from the hoist motor and the air brake will set on the drum.

A Webb trip recorder, with belt drive from the drum shaft, gives a 24-hr. record of hoisting. It records graphically the time consumed for each hoist and the time between hoists. It is arranged to indicate when inspections of the cages have been made and also may be connected to show whether or not signals have been given from the landings.

Temporary Lamp House

One corner of the substation is temporarily fitted as a lamp house. Charging racks and equipment for maintenance and repair of the miners' Edison electric cap lamps have been installed as well as equipment for the care of Koehler flame safety lamps.

Safeguards

An inspection of the substation will show that little expense has been spared to make the equipment safe to work about. Heavy rubber mats before the switchboard and around the motor-generator sets reduce the hazard of electric shock to the station attendant. Entrances to the back of the main switchboard and hoist-control switchboard are closed with expanded-metal gratings. Exposed revolving parts and terminals of the motor-generator sets are guarded. The hoist is protected in much the same manner. Automatic safety gates close the cage landings as soon as the cages leave the landings.

Fan House

The fan house is a roomy pressed-brick building in which is installed a 200-hp., 2200-volt, variable-speed, induction motor belted to a large ventilating fan. It is the intention to replace the 200-hp. motor with one of 300 hp., when the mine is more fully developed. A fuel-oil engine has been provided; by means of clutches a quick change may be made from motor to engine drive in case of failure of power or damage to the motor. The belts, rotating parts, and back of the fan-starting panel are enclosed by railings or expanded-metal guards.

Alternating current at 2300 volts is supplied to the main hoist building by a transmission line anchored at one end to the transformer substation tower and terminating at a wooden pole just outside the hoist building. The circuit is carried down the pole and into the building in conduit and branches out to two sets of three disconnecting switches. The transmission line has protection against lightning by the use of Peirce "Universal" lightning arrestors.

Main Hoist

The main hoist is the first Ilgner-Ward-Leonard hoisting installation in the Pittsburgh district, having been in productive operation since the second week of last April. Although the advantages of the Ilgner-Ward-Leonard system of electric hoisting have been known for a long time and a large number of hoists operating on this system have been installed in this country, its application is much more extensive abroad, particularly in continental Europe.

In the Ward-Leonard system, the winding drum is driven by a shunt-wound direct-current motor, either direct-connected or geared to the drum. The hoist motor receives power from a separate shunt-wound generator, to which it is direct connected electrically, the generator is usually driven by an alternating-current motor of the wound-rotor induction type. Excitation for the fields of the hoist motor and generator is supplied by a low-voltage exciter, forming part of the motor-generator.

By varying the field strength of the separately excited generator, and thereby the voltage applied to the hoist motor, the speed of the motor is controlled, the motor field has full strength throughout the operating cycle. By reversing the field of the generator, its polarity is reversed and consequently the direction of rotation of the hoist motor. As the control is affected by variation of the generator-field current, which even at the lower exciting voltage is very small compared to the main armature currents, the heavy rheostatic losses that occur during the accelerating and retarding periods with other forms of control are avoided. With hoists, such as the one under discussion, designed to operate continuously on a fast cycle, the accelerating and retarding periods make up a large part of the time and reduction in losses during these periods result in considerable economy in the daily power consumption even when considering the added losses in the motor generator.

By the addition of a flywheel to the motor-generator set and an automatic regulating device to vary the speed of the wound-rotor induction motor driving the set, the heavy peak loads taken by the hoist can be reduced or fully equalized, resulting in a practically uniform demand. The addition of the flywheel to the motor-generator set produces what is known as the Ilgner-Ward-Leonard system. In practically every instance, the motor-generator set is designed for full equalization, the few exceptions being those cases in which the operating cycle is of such long duration as to require an excessively heavy flywheel.

On any one point of the controller, the speed-torque curve of the hoist motor shows a practically uniform and comparatively small change in speed over a wide range in load, either motoring or regenerating. The superiority of the Ward-Leonard system over the induction motor for

hoists is largely due to this characteristic. Smooth and efficient electric braking is obtained by reducing the generator voltage, causing the motor to regenerate and drive the generator of the set as a motor, storing energy in the flywheel or returning it to the line. Complete control of the motor is thus obtained throughout the operating cycle and the mechanical brakes are little used except for holding the hoist at rest.

In general, the Ilgner-Ward-Leonard system is used for those hoists that operate at such speeds as to require a highly accurate and reliable form of control, or where the resulting economy warrants the additional investment. In many cases, the high peaks would so disturb the operation of the other apparatus operating from the same system, or result in increased charges when power is purchased under rates penalizing peak loads, that the addition of equalizing equipment is necessary.

Duty

The operating conditions of the Coverdale hoist, on the basis of which the equipment was designed, are as follows:

Total lift	405 ft.
Inclination of shaft to horizontal	90°
Weight of cage	20,000 lb.
Weight of load (coal)	10,000 lb.
Weight of load (slate)	16,000 lb.
Weight of cars	5,000 lb.
Diameter of rope	1¾ in.
Weight of rope per foot	4.85 lb
Type of drum	Single cylindro-conical
Diameter of drum	9 to 11 ft.
WR^2 of rotating hoist parts	750,000 lb.-ft.^{2}
Maximum drum speed	45 r.p.m.
Operation	Balanced

The cycles of speed and motor output calculated from these conditions, with an additional curve of the motor output obtained when hoisting the heavier load of slate, are shown by Fig. 1. As the controller is provided with automatic relays that limit the current taken by the hoist motor during the accelerating period, the same peaks are obtained under the two conditions of loading but a longer time is required to accelerate the heavier load to full speed.

This hoist is used only for handling coal and slate, an auxiliary induction-motor hoist handles men and supplies. The equipment is designed for continuous operation in accordance with these cycles with a short interval for caging, and to insure continuity of service a temperature rise of 40° was guaranteed for all the main electrical equipment. A slip regulator will hold the power input at a practically constant value when adjusted for the service; the power output required varies considerably

with the tonnage hoisted. When hoisting coal continuously at the rate of 600 tons per hour, the input from the line will be approximately 550 hp. The estimated over-all efficiency under this condition is approximately 44.5 per cent. and the power consumption 0.69 kw.-hr. per ton.

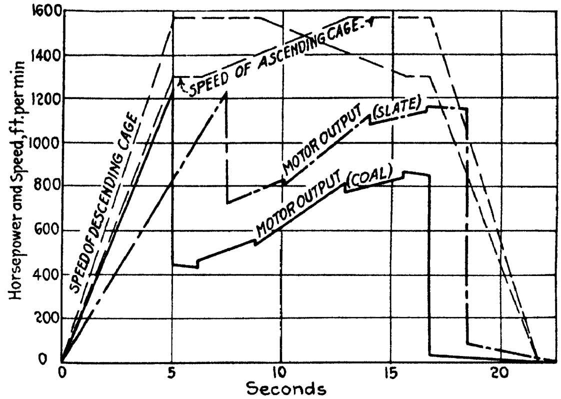


FIG. 1.—CYCLES OF SPEED AND HOISTING-MOTOR OUTPUT.

Description

The main shaft is of rectangular section with semicircular ends, and is concrete-lined a distance of 340 ft. (103.6 m.) from the landing blocks to the collar. It is surmounted by a steel headframe, which forms part of the tipple structure, the additional 65 ft. (19.8 m.) to the dumping position making the total lift 405 ft. (123.4 m.). Each of the two hoisting compartments contains a Lepley self-dumping cage designed to carry two cars side by side on a single deck, the cars being held rigidly in position by wheel clamps until released as the descending cage reaches the landing blocks.

The hoist is of the end-lift type, the equipment being installed in a substantial brick building of two rooms located on a line with the long dimension of the shaft. One room contains the hoist; the other, which is of equal size, contains the contractor panel, flywheel motor-generator set, slip regulator, switchboard, and auxiliaries.

The hoist drum is of the cylindro-conical type, having two cylindrical sections 9 ft. (2.7 m.) in diameter, two conical sections, and one 11-ft. (3.4 m.) cylindrical section in the middle. The drum is cast in two sections, bolted together at the middle, the plane of the split being at right angles to the shaft. The surface of the drum has machined grooves for 2.84 active turns on each of the small cylinders, together with the neccessary number of holding turns, five turns on each conical section, and 4.84 active turns on the large cylinder; part of this section is used

for both ropes. A brake tread is bolted to each end of the drum, provided with parallel-motion post-type brakes, which are set by weights suspended in guides beneath the floor. One of these brakes is for emergency use only; it is released by a hand lever and is held by a rachet during operation. The other brake is released by an air cylinder, served by a motor-driven air compressor, and is controlled by a second lever on the hoist platform. A normally excited solenoid is used to release a weight and set this brake in case the solenoid circuit is broken by any of the protective devices. The brake is also provided with a switch to compel adjustment as it becomes slack by the wearing down of the lining.

The drum is driven through single reduction herringbone gears, the gear unit being enclosed in a steel case. The drum and gear are mounted on a single shaft supported on three split pedestal bearings with oil-ring lubrication. The pinion shaft is carried in three similar bearings, an extension of this shaft being provided with a small band brake. A Francke flexible coupling is used to couple the motor to the pinion shaft. The complete hoist, including the motor, is mounted on a single cast-iron bedplate.

The hoist is driven by an 850-hp., 350-r.p.m., 600-volt, shunt-wound, direct-current motor. The motor has eight main poles with coils wound for 250-volt excitation, and eight commutating poles, insuring excellent commutation at the overloads obtained during acceleration. The armature rotates in two split pedestal bearings provided with oil-ring lubrication. The armature has a WR^2 of 16,000 lb.-ft. The frame is of cast steel in two sections bolted together on a horizontal diameter, the height of the motor being 5 ft. 8⅝ in. (1.7 m.) and the total weight 23,000 lb. (10,433 kg.).

The hoist motor receives power through a flywheel motor-generator set consisting of a direct-current generator, exciter, and flywheel driven by a wound-rotor induction motor connected to the 2200-volt line. The generator of this set has a continuous rating of 750 kw., 880/790 r.p.m., 600 volts, and is provided with six main poles with coils wound for 250-volt excitation and six commutating poles. As this machine must commutate the overloads during the accelerating period with a field varying from zero, at the beginning, to a maximum, at the end, a compensating winding is placed in slots in the face of the main poles insuring excellent performances under this severe condition.

The induction motor driving the set is rated 500 hp., three phase, 60 cycles, and has an eight pole winding with a synchronous speed of 900 r.p.m. This motor is of the wound-rotor type with a three-phase secondary connected to the slip regulator to limit automatically the input to the set.

The flywheel, located between the motor and the generator, has a

weight of 20,000 lb. (9072 kg.) and a diameter of 8 ft. (2.4 m.), so that the peripheral speed at synchronism is 22,600 ft. per min. (10,251 m.). The drop in speed at the end of the coal-hoisting cycle is approximately 9 per cent. and 15 per cent. when hoisting slate. The wheel is constructed of ½-in. rolled steel plate riveted together between 1-in. end plates and machined over the entire surface. A reinforced steel case encloses it above the bedplate to protect attendants from injury and to reduce windage.

The exciter armature is bolted to an extension of the generator shaft, the frame being carried on a pedestal forming part of the bedplate. The exciter has a continuous rating of 12.5 kw., 250 volts, supplying excitation to the fields of the hoist motor, the generator, and the contactor coils of the controller, the latter requiring a comparatively small amount of power. An overspeed device is mounted on the induction-motor end and arranged to trip the main coil circuit breaker if the speed of the set rises very much above synchronism, although this protection is hardly necessary in the present case.

The rotating elements of the motor-generator set are mounted on three sections of shaft carried in four split-pedestal bearings, two of which are located at the outside ends of the motor and generator, the remaining two supporting the flywheel at the middle of the set. The small exciter shaft is bolted at one end to the generator shaft and carried at the outside in a bracket bearing. The flywheel is mounted on a short section of shaft solidly coupled outside of its two bearings to the shafts of the motor and generator, the half-couplings being forged integral with the shafts.

The four pedestal bearings have split cast-iron shells lined with babbit and provided with both gravity feed and oil-ring lubrication. The bearings are fed from a gravity tank located about 15 ft. (4.6 m.) above floor level, the overflow from the bearings being piped to a tank below the floor, where it is filtered and then returned to the gravity tank by a small motor-driven pump. In addition to gravity feed, the shells of the two flywheel bearings are cored out and supplied with cooling water obtained from a pressure and cooling tower outside the building to which the warm water is returned by a small centrifugal pump. This system also supplies circulating water to the cooling coils of the liquid slip regulator.

The entire set is supported on a single cast-iron bedplate on a concrete foundation of sufficient size and depth to insure permanent alignment. The pedestal at the induction-motor end of the set is carried on a removable bridge bolted to the bedplate to allow the stator to be shifted, if necessary to inspect the windings. The complete motor-generator set is 21 ft. 9¾ in. (6.7 m.) long, 9 ft. (2.7 m.) wide, 6 ft. 2 in. (1.9 m.) high, and weighs 66,850 pounds.

The set was guaranteed to start from rest with an input not exceeding

$1\frac{1}{4}$ times the full load rating of the induction motor (500 hp.) and in service starts at slightly above this full-load value. In case it is desirable to decrease this demand, provision has been made in the flywheel bearings for connection to a high-pressure oil pump to supply oil pressure under the shaft before starting. The input to the set running light at full speed with no load on the exciter is approximately 50 kw.

The collector rings of the induction-motor driving the set are connected electrically to the three terminals of an automatic slip regulator adjusted to limit the input to the set to the proper value. The slip regulator is essentially an adjustable three-phase liquid rheostat in the secondary of the motor, the resistance of which is automatically varied within limits when the current input to the set tends to change from the value for which the regulator is adjusted. The regulator consists of a rectangular iron tank with three insulating wooden pots suspended from the bottom, the tank and pots being filled with an electrolyte made up of a solution of sodium carbonate (soda ash) in water.

In the bottom of each pot is a stationary plate, or electrode, connected through a terminal on the outside to one of the collector rings of the motor. The terminals are made in the form of valves for draining the tank when necessary, a pit being provided under the regulator to receive the electrolyte. Three movable plates suspended in the pots from a horizontal steel bar, which serves also as a common electrical connection, form the star point of the adjustable three-phase resistor. The movable plates are suspended from one end of an arm carried on the shaft of a torque motor mounted on top of the tank, the weight of the plates being partly counterbalanced.

The torque motor is a small, three-phase, wound-rotor motor without slip rings, designed to rotate through a small angle, the primary of the motor receiving power from current transformers in the alternating-current line to the motor-generator set. The torque of this motor accordingly varies with the square of the line current and when this current tends to exceed a predetermined value, the torque overcomes the unbalanced weight of the movable electrodes increasing the separations between movable and stationary electrodes. This introduces resistance in the secondary of the main induction motor, which increases its slip, allowing the flywheel to slow down and assist the motor in supplying power to the generator. When the current tends to fall below the predetermined value, the torque motor will not be able to hold the unbalanced weight and the resistance will automatically decrease, energy being stored in the flywheel as the set speeds up. When the set is running at full speed and the hoist is idle, the movable and stationary plates are practically together, the resistance is low, and the losses light. Cooling coils are immersed in the electrolyte through which water is circulated to carry away the heat developed during normal operation.

The regulator is adjusted to operate at any desired current by means of four taps (80 to 150 per cent.) on the secondaries of the current transformers supplying the torque motor and by weights counterbalancing the movable plates, permitting adjustment for low input during development. When adjusted for the proper value, as determined by the service, the regulator will hold the demand to a practically uniform value.

The slip regulator is also used as a liquid rheostat to start up the set, and is provided with a low voltage release and a contact connected in the circuit of the lockout coil on the main oil circuit breaker, which prevents starting the set until the plates are at maximum separation.

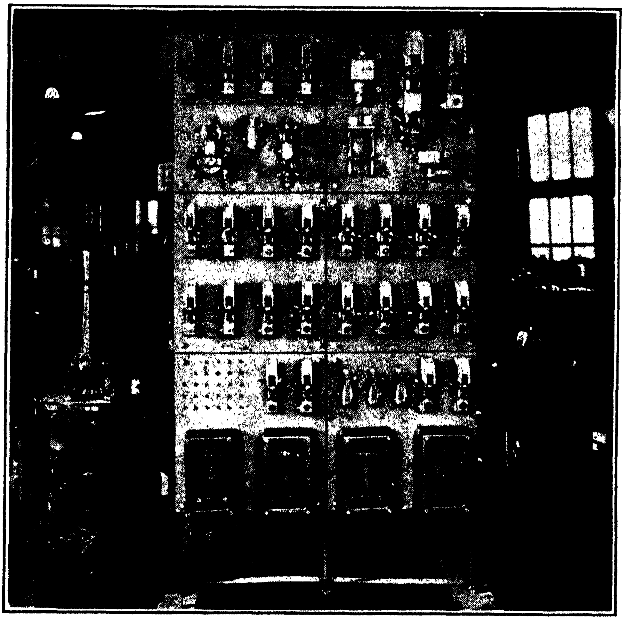


FIG. 2.—MAIN-HOIST MOTOR-CONTROL EQUIPMENT.

The control equipment consists, essentially, of a vertical-handle reversing-drum master controller on the hoist platform and a small magnetic contactor panel located in the motor-generator room, as shown in Fig. 2. By means of this equipment complete control of the hoist motor is obtained by varying the field of the direct-current generator, the armatures of the two machines being connected solidly together electrically, with only an overload relay in the armature circuit. The wiring diagram of the controller is shown in Fig. 3.

The contactor panel includes a two-pole exciter circuit breaker, four mechanically and electrically interlocked reversing contactors for

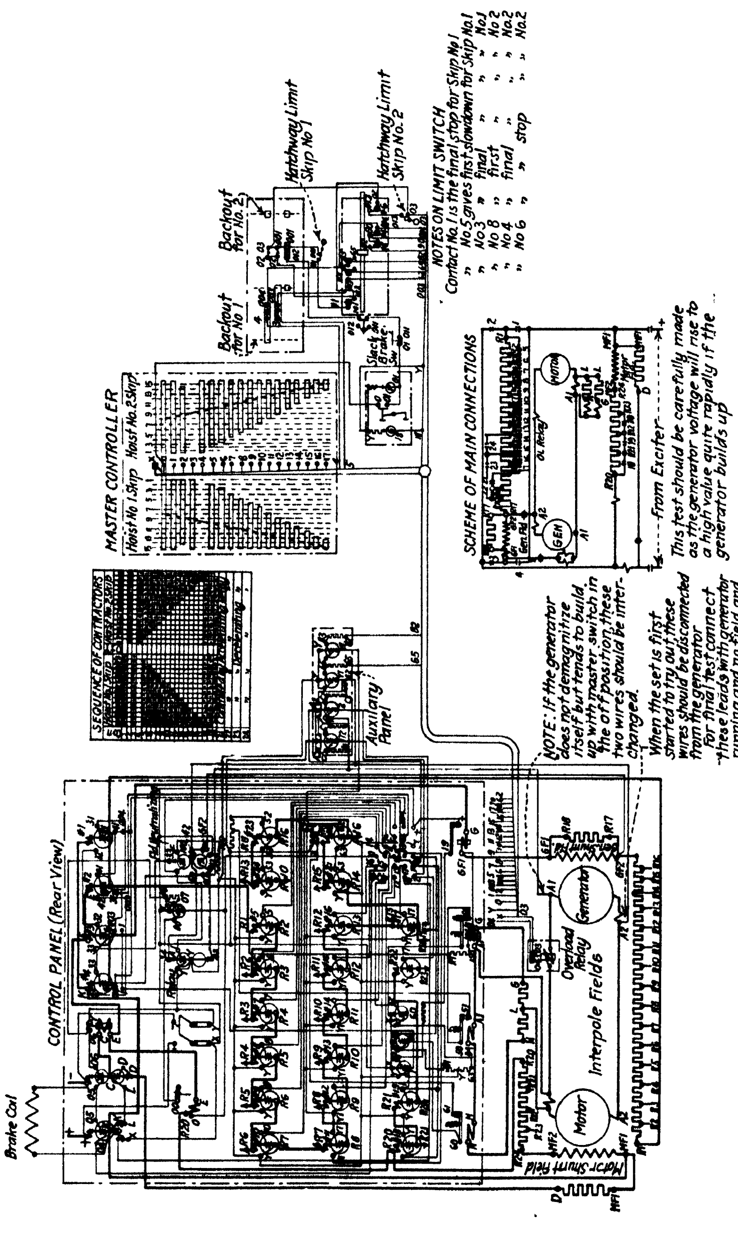


FIG. 3.—WIRING DIAGRAM FOR MAIN-HOIST MOTOR.

the generator field, accelerating contactors to increase the strength of the generator field, motor-field weakening contactors, automatic current-limiting relays, and the necessary interlocking relays. An exciter overload relay is mounted on the panel while the main overload relay is located on a small separate slate.

The hoist cycle is started with maximum motor field and weak generator field; the first point of the master controller closes one set of generator-field reversing contactors and each successive point one of the accelerating switches. If the master controller is moved suddenly to the full on-position, an automatic accelerating relay of the Tirrel type with contactors working on a separate section of resistance in the generator-field circuit prevent too rapid an increase of field strength and limit the accelerating current to the proper value. Reduced speed operation is obtained on any point of the master controller, fifteen speed points being provided. The motor field is forced during the accelerating period, a contactor opening when the motor is about up to speed providing one point of motor field control. To reduce heating, the motor field is weakened in the off-position of the controller, the field remaining partly excited, however, as long as the exciter circuit breaker is closed. When first starting up after this circuit breaker has been opened, a small relay prevents exciting the generator field until the motor field has increased to a safe value.

To retard the hoist, the master controller is moved to the off-position, decreasing the generator field and providing regenerative braking. Separate relays similar to the accelerating relays prevent too rapid decline of generator field and are adjusted to hold the retarding current to the proper value. The current-limiting relays effectively prevent overloading the equipment if the master controller is suddenly moved to the full on-position, returned to the off-position, or reversed. This makes a reliable and satisfactory form of controller that requires little attention and effort on the part of the hoist operator, obviously of great advantage on hoists, such as the one under discussion, having cycles of short duration that are rapidly repeated throughout the day.

A traveling-nut limit switch driven by the hoist provides two automatic slow-down points in each direction of travel, opening the accelerating contactors in two sections. This switch is also provided with limit contacts and an overspeed device that opens the exciter breaker and sets the brake in case the hoist is allowed to pass the limit of travel or overspeed. The limit switch is supplemented by switches of the track type installed on the headframe and operated by the cages. A small drum back-out switch mounted on the hoist platform permits operation only in the reverse direction in case of overwind. An emergency-stop button is also provided by means of which the operator can instantly open the exciter circuit breaker and set the brake.

The switchboard equipment consists of an incoming alternating-current line panel and an exciter panel located near the motor-generator set and a small panel on the hoist platform with a double-scale zero-center direct-current ammeter and voltmeter. The alternating-current panel includes the necessary meters and a double-throw oil circuit breaker for connecting the induction motor to the line and for plugging the motor in case it is necessary to stop the motor-generator set quickly. The exciter panel includes an ammeter and voltmeter and a voltage regulator to hold the exciter voltage constant as the motor-generator set drops in speed.

The same careful attention to safety features that was given to equipment in the substation is also found in the guards and railings used in the main hoist building.

Tipple

The tipple is of substantial steel and concrete fireproof construction. The self-dumping cages are a little out of the ordinary in that they accommodate two pit cars of 5000 lb. capacity side by side instead of only one. After being dumped, the coal passes into weigh pans under the weigh house at the top of the tipple. From these it goes to the shaker screens, or it may be diverted into the mine-run chute.

Another departure is the use of circular picking tables instead of the straightaway type; this type of table permits of a compact arrangement of equipment in the tipple. When slate and rock are hoisted, air-operated gates in the top of the tipple direct them into a bin whence they are taken to the dump by an electrically driven larry.

Three 37½ kv.-a. transformers in the generator room of the hoist building furnish alternating current at 220 volts to the tipple for power purposes. There are two 50-hp. motors in the tipple; one drives the shaker screens and the other is connected to the picking tables and loading booms. Each of the loading booms has a 5-hp. motor for raising and lowering. Besides, one 6.8-hp. motor operates an air compressor and a 5-hp. motor the conveyor that carries the picking-table refuse to the rock bin. As far as practicable all wiring in the tipple is enclosed in rigid iron conduit and there are no open switches for the men to handle.

Signal System

A rather complete signal system is installed in the tipple and connects with the hoist room and shaft bottom. For signaling and communication from the shaft bottom to the hoisting engineer and the top man, the familiar pneumatic signal is used. The top man signals the hoisting engineer for cage movements by a push-button operated electric bell.

When rock is to be hoisted, the man at the shaft bottom presses one or two push buttons corresponding to the location on the cages of pit cars with rock. A bell in the top of the tipple rings and a red light shows over each car carrying rock until the cage is dumped. This signal is designed to warn the top man in ample time so that he may throw the fly gates into the proper position and thus avoid sending rock into the weigh pans or coal into the rock bin. A klaxon horn sounds at the same time that the bell rings, so that the tipple foreman will be on the alert should any rock get through the weigh pans.

Another system of signals with push-button operated klaxons enables the man on the loading platforms to signal for stopping and starting of loading machinery, shaker screens, etc. when shifting cars or for any other reason.

An important factor in deciding the type of equipment for the main hoist was the increase in generating and transmission equipment that would have been required to operate the hoist by an induction motor. This, together with the operating advantages of the Ilgner-Ward-Leonard system, led to the adoption of the hoisting equipment described. At the present time, the mine has not been developed to a sufficient extent to work the main hoist to capacity, but in ease and accuracy of control the equipment is demonstrating its superiority.

Automatic Substations Used in Coal Mining

By R. J. Wensley,* East Pittsburgh, Pa.

(Wilkes-Barre Meeting, September, 1921)

The use of small substations for the supplying of 275-volt energy to the locomotive and cutting machines in coal mines is a well-established practice. A few years ago, when labor costs were lower, these substations were located as near to the load as possible, and an operator was provided for each station. This practice has now become so expensive that substations are being located with reference to other mining machinery, such as hoists or pumps, so that one operator can look after both. Sometimes the substation is operated by one whose duties make him traverse quite a large area. In such cases the interruption of alternating-current supply may produce a long interruption in the direct current, as the station cannot resume service until this man reaches it. Such interruptions interfere seriously with production and may easily counterbalance the supposed saving gained by such operating methods.

The grouping of substations for convenience in operating may also result in excessive copper loss and consequently poor trolley voltage, with its attendant evils of low locomotive speeds and increased locomotive motor maintenance. The speed of the coal-cutting machines will also be reduced. By relieving the substation of its burden of the operating labor cost, the most economical location, from an electrical standpoint, may be chosen.

The matter of machine insurance should also be considered. If an attendant is caring for several kinds of equipment, he cannot watch the machine continually for signs of trouble; therefore, continued overload, phase failure, low alternating-current voltage, bearing trouble, etc. may result. Some partly automatic equipments have been installed; these have protection against low voltage, reverse current, and bearing trouble and are provided with automatic reclosing circuit breakers. This is a step in the right direction but it does not go far enough to give first-class machine insurance.

The saving in maintenance costs is often the smallest item, when considering the automatic substation as machine insurance. When the coils in a machine are roasted out, due to continued overload or to opera-

* Switchboard Engineering Dept., Westinghouse Elec. & Mfg. Co.

tion under abnormal alternating-current conditions, the loss in production will easily offset the repair bill in most instances. Automatic control allows the installation of the substation above ground, when the mine is not too deep, but the location depends largely on local conditions. In some cases the topography of the country is such that an inside station would be much more accessible than an outside one, and vice versa. The most economical method of getting current into the mine is by the use of overhead lines and drill holes directly above the desired point of feed. If the substation is installed in the mine, there will be a triplex cable in the drill hole usually working at 2200 volts, although a few mines use higher voltages. If the substation is installed on the surface, the direct-current cable is placed in the hole.

The surface location has many advantages, one being the lower installation cost. Inside installations usually require concrete and steel work and in many cases considerable excavation of rock. The substation location must be such as to receive an ample supply of cool fresh air. The control wiring must be carefully done and usually with lead-covered cable, for sulfur water is often present. The installation of the machine is frequently difficult as it must be dismantled so that the largest piece can be handled by the hoist or to pass through the available openings. If the substation is placed on the surface, an inexpensive house may be used and the machine may be set on the floor just as it comes from the factory. There will be no dampness to cause deterioration of the equipment, and no ventilation difficulties; also expensive runs of high-tension cable are avoided as the lines may be run overhead.

Remote-control Operation

The most convenient method of automatic operation is by remote control through the high-tension feeder. For the most reliable operation, it is necessary that each substation have an independent feeder. Then, the stations may be started by closing the oil switch at the point of origin of the feeder; an ammeter installed at this point will show the attendant what is happening at the substation. If to reduce the first cost it is necessary to put more than one station on one feeder, a series of time-element relays can be used to start the machines at short intervals, so as to avoid the surge otherwise caused by simultaneously closing the starting switches of several sets. The stations may also be started by voltage relays connected to the trolleys and shut down by under-current relays in the machine circuits. This is a more expensive method and is warranted only when the power cost is high. Time switches may be used to start the stations according to a definite schedule if desired. Control may be had from pilot wires run to convenient points where attendants are always available. The simplest method is the use of a

starting button at the door of the substation, to be operated by the first man in and shut down by the last man out. This method, while inexpensive, is cumbersome and not to be recommended except in special cases.

Designing Mine Automatic Equipment

In designing automatic equipment for mining service, the manufacturer must take into account the usual lack of technical training in the maintenance crew and must seek extreme simplicity. The installation of a mining substation is usually of a semipermanent character and operators are reluctant to install expensive or complicated equipment that, in a relatively few years, must be moved to a new location or scrapped. At first the tendency of the manufacturers was to offer an elaborate equipment of the railway type modified only as to voltage; later, as the operating conditions were not as severe and the standard of service was not as high as in the railway field, a special line of automatic control was developed for mining service.

This equipment, for the control of a 200-kw. synchronous motor-generator set is on two panels. The wiring is self-contained so as to avoid the necessity of complex control circuits apart from the panels. The only wiring, apart from the main motor and generator leads, is the control circuit from the control transformer and the bearing thermostat connections. The panels are wired at the factory so as to reduce the installation cost to a minimum and to avoid errors. This equipment is designed to operate on an individual high-tension feeder; it starts whenever alternating-current is thrown on the line and is shut down by opening the alternating-current feeder. The direct-current generator is connected to the line through an automatic reclosing circuit breaker, which opens on short circuit or excessive overload and recloses when the cause of the trouble is removed. This may be when a broken trolley wire is picked up or removed from the rails or the locomotive operators throw off their controllers. It is possible to substitute, for this breaker, the standard railway equipment, consisting of a number of contactors each short circuiting a portion of a current-limiting resistance. These contactors are arranged to open one at a time until the overload is limited to within the commutating capacity of the machine. By using this scheme, the locomotives are kept in motion at reduced speed until the current drops below the setting of the overload relays, when the contactors will again close, thus cutting the resistors out of circuit. This equipment costs somewhat more than the type using the automatic reclosing breaker and therefore will not be in as great demand.

In large operations, the best service insurance is obtained by the use of a series of automatic section switches to isolate portions of the trolley, thus allowing the remainder of the operation to proceed. Where feasible,

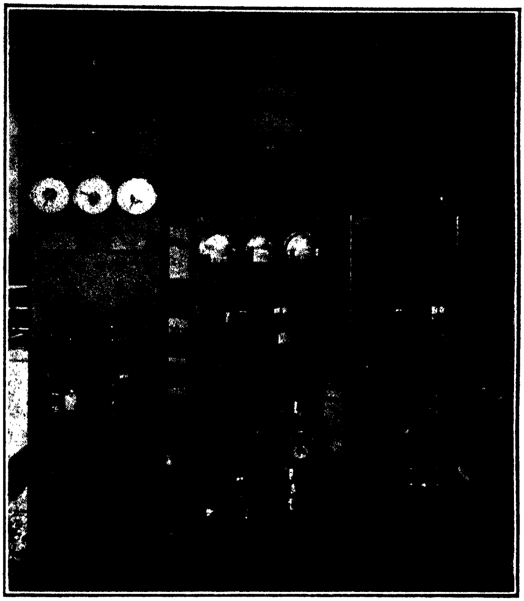
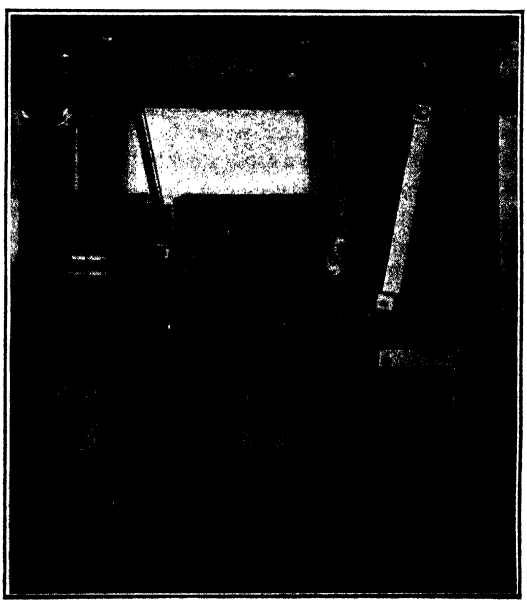


FIG. 1.—FRONT OF CONTROL EQUIPMENT OF AUTOMATIC SUBSTATION.



IG. 2.—REAR OF CONTROL EQUIPMENT OF AUTOMATIC SUBSTATION.

additional reliability may be insured by running separate direct-current feeders from the substations for the coal-cutting machines; this is to prevent stoppage of the actual mining operations in any given section in case of trouble with the trolley.

Cost of Automatically Controlled Substations

There is a feeling among coal operators that automatically controlled substations are too expensive for the average mine. This is not at all true. A 200-kw. set with 2200-volt 60-cycle synchronous motor and 275-volt direct-current generator with manual switching would sell at the present time for approximately $6150. The same machine with the simplest type of automatic control will sell for approximately $8350. This difference of $2200 capitalized at 20 per cent. per year will give an annual charge of $440, which is far less than the wages of a man to operate the station for even one shift per day. If the operating labor is kept below this point by giving only occasional attention to the station, serious interruptions are invited as previously mentioned. But in addition to the saving in wages the automatic station does the right thing at the right time much better than it could possibly be done by manual operation.

First Automatically Controlled Substation

The first coal mine to adopt automatic control for substation equipment is that of the Lincoln Coal Co. at Nant-y-Glo, Cambria County, Pa. This is a 200-kw. synchronous motor-generator set located in a room about 2 miles from the mouth of the drift and about 500 ft. under the surface. The 2200-volt three-phase power is supplied from the surface through a 5-in. drill hole in which a triplex cable is installed. The station is started by the first locomotive runner in the morning and is shut down by the last man trip out at night. This is accomplished by a single small knife switch. It operates continually during the day. If alternating-current should fail, it will shut down but immediately after the alternating current service is restored, the station will start and go on the line in a few seconds. This one feature alone, in a busy mine, warrants the slight extra investment in the automatic equipment.

Figs. 1 and 2 illustrate the front and rear of the control equipment of an automatic substation installed during the summer of 1921.

DISCUSSION

W. B. Flynn,† Philadelphia, Pa.—On one of the properties in which the Day & Zimmermann Co. is interested, the automatic stations give considerably less trouble than the manual-controlled stations, doubtless

† Day & Zimmerman, Inc.

because of the elimination of the personal element. In the case of substations we are erecting for a traction road in Ohio, the manual-controlled stations cost about $22 per kilowatt of installed rotary converters, while the automatically controlled stations cost about $32 per kilowatt of installed converters. The saving of wages of station operators will of course more than offset the interest on this additional investment."

GRAHAM BRIGHT, East Pittsburgh, Pa.—This paper covers largely the automatic substation for mining, but automatic application of electrical equipment will no doubt go further than the substation. At one of the Lehigh Valley mines, some of us saw today an automatic fan equipment which has recently been installed. In case power goes off, the fan shuts down but as soon as power comes on again the fan automatically starts up, requiring no attention except an inspection possibly once a week. There are other applications, such as pumps, compressors, etc. that could well be operated by automatic control equipment, so that in the near future we can expect to see some rapid advances made in the application of automatic equipment in other locations than the substation itself.

EDWIN LUDLOW, New York, N. Y.—At a quarry of the Bessemer Limestone Co. near Youngstown, Ohio, 1400 tons of rock were conveyed daily from the quarry to the crusher by four different power cars operated by a man standing in a tower. These cars traveled up and down hill, went into a siding, stopped, etc. entirely under the control of this one man.

Electric Power a Factor in the Anthracite Field

By W. A. Thomas, Scranton, Pa.

Wilkes-Barre Meeting, September, 1921)

Steam is, and doubtless always will be, the basic power in the anthracite industry, either directly applied through engines and pumps or electrically. The rapidity with which electric power is being applied to the mining and preparation of anthracite leaves little doubt as to its utility and economy. There is practically no operation, except possibly the drilling of hard rock, that has not been successfully electrified and the matter of drilling is handled economically with electrically driven compressors and the standard compressed-air drills.

Conservation and the seemingly ever-increasing cost of production, justify careful consideration of the possible results of the more extensive use of electric power in the industry. It is not planned to discuss, herein, the relative economies of individual applications but to consider the relative economy and flexibility of steam and electric power.

The reports available for the year 1920 indicate a production of slightly over 80,000,000 tons of anthracite, of which 8,843,500 tons were used for power and heating purposes; in other words, over 11 per cent. of the production was used at the mines. Some of this was used to generate electric power at the mines, about 160,000,000 kw-hr., while 150,000,000 kw.-hr. was purchased from the public utility companies.

One large producer, who is buying some electric power and generating some, used over 13 per cent. of the production for power and heating at the mines. Another producer, where considerable electric power is generated on the properties, is using less than 9 per cent. of the output for power and heating. One of the largest producers has a fuel consumption of over 15 per cent. of the output, besides which considerable power is purchased.

A careful study of the field indicates that steam-operated plants use upwards of 13 per cent. of the production for power and heating, while electrically operated plants use about 1 per cent. for heating; so that, for steam-power purposes at least 12 per cent., or 268.8 lb., are used for power out of each ton produced.

In the complete electrification of anthracite mining operations, no set rule can be given for the power requirement, in kilowatt-hours per ton, as conditions vary widely in the matter of pumping, ventilation, and transportation, but the following cases are given as a basis of determining a range and a possible average power requirements:

Case A.—Middle field; annual production 600,000 tons; very little pumping and power ventilation; completely electrified, except stripping shovels and locomotives; power consumption, 5 kw.-hr. per ton.

Case B.—Northern field; production 525,000 tons per year; pumping and power ventilation conditions slightly below the average; complete electrification except outside haulage; power consumption, 12 kw.-hr. per ton.

Case C.—Northern field; annual production 440,000 tons; average conditions as to pumping and ventilation; completely electrified; power consumption, 12 kw.-hr. per ton.

Case D.—Northern field; annual production 380,000 tons; heavy pumping and ventilation; estimated consumption after complete electrification, 27 kw.-hr. per ton.

Case E.—Middle field; group of operations aggregating 1,000,000 tons output per year; very heavy pumping and ventilation; power consumption, with complete electrification, is estimated to be 27.2 kw.-hr. per ton.

With a minimum of 5 kw.-hr. per ton and a possible maximum of 30 kw.-hr. per ton, it would appear that 20 kw.-hr. per ton would be a conservative average value for electrified operations.

The large central station consumes from 3 to 4½ lb. of No. 3 Buckwheat coal per kilowatt-hour at the switchboard, so 5 lb. per kilowatt-hour at the mine metering point will more than cover the central-station requirement for delivered power. With 20 kw.-hr. per ton, this gives us 100 lb. of fuel per ton production at electrified operations, as against 268.8 lb. per ton at steam operations, thus effecting a possible saving of 168.8 lb. per ton produced. It is assumed that 50,000,000 tons production may be practically electrified as the power becomes available with a possible saving of 3,767,000 tons of fuel, which can be released to the market.

As a check on the above, one of the larger producers, where a large amount of electric power is generated and used on the properties, reports a total fuel consumption of 9.27 per cent. of the output.

By dividing the total kilowatt-hours generated by 20, the above determined average per ton, an equivalent electrically produced tonnage is determined. Subtracting this from the total production gives the corresponding steam produced tonnage. Multiplying the kilowatt-hours by 5 gives the fuel supposed to have been used in generating the power; and deducting this from the total fuel reported shows a remainder of fuel, which is 12.5 per cent. of the steam produced tonnage. In this case, as none of the operations are completely electrified, it is safe to assume that the electric power has been applied where it would show the greatest saving so that the general balance, thus obtained, checks closely with the values above determined.

In gaseous mines, it is not advisable to electrify the fans unless these are supplied with power from a plant on the property with reserve generator capacity or from a public utility service having two circuits to the service. This, of course, depends on the hazard involved from failure of the fan drive for short periods. For the average producer, the use of public utility power service is most economical and convenient; and if such service is available, the isolated mine operation can seldom generate its own electric power as cheaply and with as much reliability. But if the central-station lines are too remote, this and other reasons may show that a local plant would be justified. In the isolated plant, however, there should be reserve boiler and generator units to insure the same continuity of service as may be expected from the public utility company, and this makes the plant investment relatively high.

As to cost and operation before and after electrification, it is unfortunate that careful records are either not kept or are grouped with other subdivisions of production costs so that it is difficult to get many accurate comparisons of cost before and after electrification.

In case B, above noted, complete data are available. The operation had two boiler plants, both with part of the steam driving direct-current generators and one having the usual nest of air compressors. In electrifying, it was necessary to install motor drive in the breaker and washery, replace a large breaker wash-water pumping station, build three converting substations, install several new hoists and pumps, and change the drive on steam and compressed-air hoists and pumps and fans. Separate job numbers were assigned to each installation against which all charges were made for complete installation; on completion it was found that the total cost was not quite $125,000. This expenditure was made for an estimated saving of $44,600 per year; but before the year was up, 1917–18, wages, fuel value and, for part of the year, power rates had advanced to such an extent that the actual saving was over $85,000 the first year.

It would not have been possible to have operated the property under 1920 conditions with the plant equipment available in 1916 at the time of electrification; but assuming this to have been possible and that no more fuel or labor at the plants would have been necessary, a reconstruction of the 1916 cost of power on the basis of present wages and cost of supplies and the average price for fuel for the first half of this year, rather than the high value of 1920, shows a cost which, after deducting due allowance for heating is $125,000 in excess of the cost of electric power for the year 1920. The added cost for electricians, if any, is more than offset by the reduced cost of maintenance of drives.

A reconstruction of the installation cost would probably show this to be about $160,000 this year.

This installation is, perhaps, exceptional as regards the obtainable

economies but some of the gains were possible by taking advantage of changes in the mining operations, made possible by the greater flexibility of electric power distribution, such as sinking a new shaft over the head of a distant slope for ventilation and the location of slope hoist and converting substation.

It is not out of place to state that there remains on the property a water-tube boiler plant, near condensing water, which is large enough to generate sufficient electric power to operate the property, but the cost of generating, with the necessary reserve capacity, would just about equal the cost of purchased power with the additional worry of boiler repairs, arches, feedwater heaters and pumps, and the usual boiler-house organization.

In case E, cited above, a careful study shows that an expenditure. of slightly over $900,000, not taking into account a considerable amount of good equipment that could be salvaged, would effect a possible saving in operating cost of $440,000 per year at the present public utility rates for electric power. This is typical of many cases where the gross annual saving, by complete electrification, will be at least 50 per cent. of the cost of the change over and always makes possible a greater output, where this is limited by power equipment or power supply.

In this connection, it should be noted that complete electrification will show the greatest economy for the reason that, while partial electrification in many cases shows great saving, it does not reduce the total steam cost in proportion to the reduction in the steam load. The standby losses, maintenance of plant and steam lines, and the labor in the fireroom, handling of fuel and ashes is seldom reduced in the same proportion as the load.

At present, the demand for electric power in the anthracite field exceeds the supply but the public service companies are planning for increased capacity and the interconnection of systems to safeguard the continuity of service. If this is not done, many operators will doubtless install isolated or group electric power plants.

It may be argued that, with public utility power available, the isolated mine can generate electric power cheaply from non-commercial grades of fuel; while this may be true at the present time the efforts now being made to bring these grades of fuel into the domestic market are likely to change this situation. With this possibility in view, the release of upwards of 3,000,000 tons of fuel to the domestic market, becomes an important factor in the anthracite industry, both as to economy and the future domestic-fuel situation.

With a few exceptions, principally large group operations, the purchase of electric power from the well-organized public utility systems works out to advantage.

The indications point to the probable development of large public

utility plants of possibly 100,000 kw. capacity, where abundant water is available near the fuel supply, and so interconnected as to render reliable service to the entire anthracite field and, in many cases, connect with the large group operation systems.

It is not too optimistic to predict that one such plant could be loaded to capacity in the next 5 years, nor would it be too visionary to presume that some of this power may be produced from bituminous coal if the expectations of bringing small sizes of anthracite into domestic consumption be realized.

There is but one conclusion, and that has been reached by many producers; viz., electric power, when properly installed, is cheaper and much more flexible than steam in carrying on the mining and preparation of anthracite.

Emphasis should be placed on proper installation not only from an electrical engineering standpoint but from a mining and preparation standpoint as well, in order to take advantage of the possible changes in the handling of the product by reason of the greater flexibility of electric power. Due regard should be given to the conditions of the public utility rate schedules as to "load factor" and "power factor" and to the tendencies in rate making, as bearing on the eventual cost of power for any specific operation.

The piecemeal, or gradual, electrification of an operation should be carried on in conformity with a general plan as a result of a complete survey or study of the proposition as a whole. There are some advantages in carrying out an electrification in this way particularly where conditions are changing due to second mining; but where conditions are stable, the more quickly it can be done the sooner can the savings be made effective.

DISCUSSION

R. W. WILBRAHAM,* Philadelphia, Pa.—In the southern fields of West Virginia, particularly on the Coal River, in Boone and Logan Counties, all the plants are completely electrified, due to economies to be effected. Nearly all the companies operate a number of mines, which are from 1 to 12 mi. apart.

Originally power was purchased on an energy basis on the direct-current side of the motor generators; the power company supplying the substation equipment. Recently, the Public Utilities Commission approved a schedule of rates involving a demand and energy charge, the latter being on the "block" basis, *i.e.*, cost per kilowatt-hour decreased as the consumption increased into each succeeding group, or

* Day & Zimmermann, Inc.

"block." Under this schedule, consumers were obliged to own all substation equipment and to purchase alternating-current energy.

In the first month after this schedule became effective, there was an increase, in the case of one group of mines of approximately 47 per cent., caused partly by the fixing by the Commission of an arbitrarily fixed load factor to be effective only prior to the installing of proper meters by the power company, but the final increase was 20 per cent.

We then investigated the possible economies to be effected by purchasing all energy at one or two points, in place of several, as heretofore. Fortunately, there was available data on the consumption of each mine and a graphic meter study was made of the entire system to determine voltage drop and power losses. An investment of approximately $12,000 was then made to effect the necessary changes and additions for metering all energy at two points in place of five, which resulted in saving $11,000 a year. Taking the fixed charges of 12 per cent., or $1440 a year, the entire investment, including fixed charges, can be amortized from the savings realized in approximately 15 months. At the end of this period, the savings will represent a return on the investment of approximately 90 per cent.

As one of the mines of this group is on a low-production basis, on account of recently being opened, also as it is a considerable distance from the other mines in this group, it is not considered economical at this time to make the required line extension to accomplish all the metering at one point. This, however, can be accomplished at any time when the production warrants the change. If these savings can be accomplished in the soft-coal fields, they would seem to substantiate Mr. Thomas' statements as to the enormous savings that can be made in the anthracite region.

James H. Pierce,* Minersville, Pa. (written discussion).—Companies that compare the cost of purchased power with the cost of operating their antiquated steam plants make a serious error; steam, properly applied, is a close competitor of electric current, and the problem must be worked out for every individual operation. If steam is generated in a modern power plant and then fed to high-efficiency condensing turbines used for driving motors at a colliery, the cost will be less than the cost of purchased power, but often such a plant is not used on account of the heavy investment required, for the boiler plant, turbine, condensers, spray ponds, besides the full electrification investment. But once the investment is made, provided the life of the plant will warrant it, steam applied in this way has many advantages over purchased power, besides a greater degree of safety in the continuity of operation and a smaller cost at the switchboard.

Where the operation is small or the life of the lease is limited, one

* Assistant to the President, Buck Run Coal Co.

cannot afford to make so heavy an investment, and purchased power will be found much more economical than any other combination.

In one operation that was carefully investigated, boiler tests were run, every individual electrical unit was indicated, and competent electrical engineers engaged to determine whether it would pay to electrify the plant; to the amazement of all it was found that a new boiler plant with a steam turbine, was by far the more economical but when the investment, life of lease, salvage value, etc. were taken into consideration over the short life of the property, it was deemed wiser to make the smaller investment necessary in electrification, and be satisfied with a smaller annual saving.

CHARLES S. DAWSON,* Scranton, Pa. (written discussion).—The conclusion reached that electric power properly installed in anthracite mines is more economical than steam drive and that the purchase of the electric power from public-utility systems, with few exceptions, works out to advantage is evident to any one in a position to observe the comparative results.

In the past, the power question has been of small concern to the average anthracite operator, who was content to operate his own boiler plant and run steam lines here and there to steam engines wherever power was needed, giving small heed to efficiencies and costs. This was mainly because he found a small market for the coal burned in his boiler plant and the margin of profit of his operation was entirely satisfactory.

Today, he is facing an entirely different situation. His operating costs have been increasing at an alarming rate and even though coal prices have advanced materially, he has been hard-pressed to maintain his customary profit and in many cases has been unsuccessful. In consequence, production costs have come under careful scrutiny. It has taken only a casual survey in many cases to indicate the gross waste and excessive expense of steam-operated mines. In other more extensive operations, careful investigation and study have clearly shown the economies to be derived from electric power. Certainly a complete analysis of any steam-operated mine by competent and unprejudiced engineers is warranted.

Mr. Thomas estimates a possible saving of 3,767,000 tons of fuel per year by electrification. This figure seems to be well within the bounds of expectation and it would not be surprising if 5,000,000 tons or more could be saved. The releasing of this fuel to the market is important in itself, besides the fact that the additional revenue accuring to the railroads for haulage is an item of some moment.

Electrification is going forward at a rapid rate and heavy demands are

* Commercial Engineer, Scranton Electric Co.

being made and will continue to be made on the central stations for the supply of electric power. To meet these increasing demands the public-service companies are being forced to the consideration of new and much larger plants which will contain turbine units of 30,000 kw. or more capacity each. The efficiencies obtainable with such units will materially lower the generating costs and consequently the rate, other conditions remaining the same, to the consumer.

It would seem, then, that in the near future electric power should not only be a factor in the anthracite field but should be almost the universal power.

GRAHAM BRIGHT,* Pittsburgh, Pa—About three or four years ago, Mr. Hobart told of the savings effected by the electrification of one of the large steam hoists of the Lehigh Coal and Navigation Co. Mr. Thomas has gone further and included the entire electrification of a mine. These economies, however, cannot be effected unless the entire electrification takes place; to electrify piecemeal is only going part way and the full effect of electrification cannot be realized. There are at this time a large number of steam installations in the anthracite fields, the electrification of which would show large returns.

The fact that an average of 20 kw.-hr. per ton is required in mining anthracite is one indication why anthracite costs more than bituminous. The average figure for bituminous mining ranges from 2 to 5 kw.-hr. per ton.

B. H. STOCKETT,† Shenandoah, Pa.—We found that by following Mr. Thomas' advice, there was a saving of $350 a month, by using a common metering point, due to the diversity of the load bringing down our demand. In one mine, the plant being one-half electrified and one-half steam, figuring coal at about $1.50 a ton, the power cost was $0.50 a ton. When the mine was completely electrified, the cost was $0.30 a ton.

C. H. STRANGE,‡ Minersville, Pa.—We all know that during the past summer it was almost impossible to sell barley coal anywhere near its cost of production. It is also impracticable to store that coal; we do not have the available room, so the barley is run down on to a slush dam; it will cost another 50 cents to a dollar a ton to screen and load that coal from the slush dam. The question is, does it pay to purchase power when barley coal is run down on to a slush dam?

As our power plant has little current to spare and one section of the mine is isolated, we have been purchasing power for this section. We

* Engineer, Westinghouse Elec. & Mfg. Co.

† General Superintendent, Locust Mountain Coal Co.

‡ Superintendent, Pine Hill Coal Co.

cannot regulate our load so as to avoid high peaks, so the demand charge has been as great as the money we pay for the energy consumed for the month. When we started, the power company offered a sort of half rate, or off-peak load, rate that was quite attractive. But just as soon as all the mines in our region were electrified, the peak load came in about 11 o'clock in the morning. So the power company said, "Our peak load is at 11 o'clock in the morning; there is not any off peak any more, so you will have to pay the full rate."

J. H. ALLPORT,* Barnesboro, Pa.—At one mine in the anthracite field, an examination showed that the power cost of producing coal and putting it on the cars was 86 cents a ton by steam operation. Tests on the steam lines showed great inefficiency and the owners of the property objected to investing in electrical equipment because the machinery was perfectly good, though it had been obsolete 25 years. After some persuasion they allowed an expenditure of $80,000. After the operation was electrified, the cost of power was reduced to 24 cents and the saving paid for the entire installation in less than 9 months.

W. A. THOMAS.—In some places, individual operators have mines where it has been possible to group the plants, but where the operations are widely separated. The question of right of way comes in to a greater extent in the anthracite field than it does in southern or western Virginia.

Fuel costs, of course, vary under different operations. In the particular case that I cited, giving figures of returns, the actual realization of the fuel previously used enters into the saving. Barley coal going into the slush pile has an intrinsic value that will eventually be recovered.

Demand charges are a stumbling block to electrification and some of the power companies have raised their charges quite high. In one case where the aggregate demand on the system exceeds the capacity of the system by 100 per cent., these demands apply practically continuously. That is one factor in rate making that should be changed; it works a hardship for operators who have a high demand for one month and a low demand another.

I wish to present on behalf of R. E. Hobart, who is unable to be present, the following figures from the Rahn Colliery:

* Consulting Engineer.

RAHN COLLIERY INCLUDING FOSTER'S TUNNEL

Average Number of Kilowatt-hours per Month, and per Ton, March 1 to Sept. 1, 1921

	Average Number of Kilowatt-hours Per Month	Average Number of Kilowatt-hours Per Ton
Drainage	62,200	1.559
Breaker machinery	136,950	3.430
Transportation	98,728	2.474
Ventilation	65,813	1.649
Hoisting	40,533	1.015
Compressed air	103,955	2.606
Miscellaneous	3,391	0.085
Total	511,570	12.818

Average tonnage per month 39,904
Average gallons of water pumped per kilowatt-hour 1032

R. J. WENSLEY,* East Pittsburgh, Pa.—The Consolidated Coal Co. at Mount Olive, Ill., is electrifying all of its mines. As it was impossible to purchase power economically, the company built an alternating-current power plant near Mount Olive and will run 13,000-volt transmission lines on its own poles to the different mining properties. It has purchased twelve automatic substations, ranging from 100 to 300 kw., which will furnish all of the power for the coal-cutting machines and for haulage. The hoists, of course, will be operated with alternating current.

R. W. WILBRAHAM.—In connection with the apparently excessive demand cost of power in mining, we might say that the Colver Coal Co., as well as others, has made traffic studies of its haulage conditions, with the result that the larger part of this phase of mining operations has been scheduled, thus effecting a considerable saving in the demand charge.

* Switchboard Engineering Dept., Westinghouse Elec. & Mfg. Co.

Determination of Electrical Equipment for a Mine Hoist

By Graham Bright, E. E.,* East Pittsburgh, Pa.

(Wilkes-Barre Meeting, September, 1921)

The rapid increase in reliability, the low cost of operation, the ready application of safety devices, and the growing availability of central-station power have made the question of installing a hoist of the electric type no longer debatable. Where adequate electric power is available, there are few cases where it would be advisable to install a steam or air hoist.

The type of electrical equipment depends largely on the nature of the power supply, its capacity, and the form of the power contract under which power is purchased. In a large majority of cases, the available electric power is alternating current, three phase, 60 cycles. There is, however, considerable 25-cycle power in use; and in a few instances constant voltage direct current is the only power available for a small or an occasional medium-size hoist.

Where a new hoist is to be installed, the designer of the mechanical parts can readily take care of the maximum stresses likely to be imposed by the motor. Where the hoist is already installed and operated by steam or air and it is desired to substitute an electric motor for the engine, a careful check should be made to determine the advisability of using the old mechanical parts. In most cases it is better to install a complete new hoist than to attempt to use old mechanical parts. A steam engine has a definite maximum torque, which the hoist has been designed to withstand, while a motor may be able to deliver momentary torques beyond the strength of the mechanical parts of the hoist.

The selection of the proper type of electrical equipment can only be made after complete information is obtained and certain calculations made to determine the capacities required. There are several methods used in making hoist calculations, but no matter what method is utilized a fair degree of familiarity with hoists and hoisting conditions is necessary before an intelligent analysis and proper selection can be made.

Forms Used in Hoisting Calculations

Some of the hoist calculation methods are long and complicated. Where such calculations are seldom made, it is better to use the

* General Engineer, in charge of Mining Section, Westinghouse Electric and Mfg. Co.

longer methods; but where the calculations are made daily, short cuts can often be used to great advantage. Engineers associated with the writer, during the last few years, have felt that the various short-cut methods that had been worked up could be assembled in such a manner that any engineer working on hoisting problems could complete the necessary calculations in a short time and the results could be easily checked by another engineer. With this end in view, forms were evolved so that it was only necessary to fill in blank spaces to make a complete set of hoist calculations. These forms have been modified from time to time, in accordance with suggestions by various engineers. At the top of the form is place for the name of the customer, filing information, the date, and the characteristics of the power system. Under the heading General Data are the specifications or the information necessary before calculations can be made. This is followed by the various steps necessary to obtain the complete hoisting cycle, which require only calculations of a fairly simple character. As many of the conditions of operation cannot be approximated more closely than 3 to 5 per cent., all calculations can be made by using the slide rule.

In order to better illustrate the value of the method, examples are given showing the determination of the electrical equipment of an alternating-current hoist, a direct-current hoist with flywheel motor-generator set, and a direct-current hoist with a synchronous motor-generator set. The forms are so arranged that a single sheet 8½ by 11 in. is required for the complete calculations of an alternating-current hoist and but two sheets are required for either of the direct-current hoists. In the typical case selected, the hoist is of the vertical type using self-dumping cages.

The output of the hoist is figured at two values, the first being 2100 tons in 7 hr. and the second 3000 tons in 7 hr. These capacities are selected to show that for moderate rope speeds the alternating-current motor can be readily adapted directly to the hoist in case the power conditions are suitable. For high-speed hoisting and short cycles, the alternating-current motor applied directly to the hoist is not suitable, and a direct-current motor using the flywheel motor-generator set or synchronous motor-generator set should be selected. The daily output of a coal mine is generally estimated on the basis of getting this output in 7 hr., allowing 1 hr. for delays.

Calculations for Alternating-current Hoist Motor

Calculation sheet No. 1, Fig. 1, shows the complete calculations for hoisting 2100 tons in 7 hr. It is seldom that most of the information required under the heading General Data is obtained from the customer; the first seven items are usually given and the engineer recommends or estimates the other eleven.

Customer	A. I. M. E	Neg.	000	Date	7-1-21
Power	2200	Volts	3	Ph.	60 Cy.

General Data

Depth of shaft	500 ft.	Total rev. per trip	
Angle of slope	90°	Max. rope speed	$\frac{500}{25-5}$ = 25 ft. per sec. = 1500 ft. per min.
Net weight of load	5000 lb.	Max. drum speed	59.7 r.p.m. = 0.995 r.p.s.
Weight of cage, skip	12,000 lb	Size of rope	1⅜-in. dia. wt. = 3.65 lb. per ft.
Weight of car	2500 lb.	Size and shape of drums	8 ft 0 in. dia. cylindrical
Balanced, unbalanced	Bal.	WR^2 drums, gears, sheaves	400,000 lb.-ft.^{2}
Max. capacity per day	2100 tons	Angular velocity	$2\pi \times 0.995 = 6.25$
Trips per hour	120		
Time for caging	5 sec.		
Running time per trip	25 sec.		
Total time per trip	30 sec.		

Time

Acceleration	5 sec.	Retardation	5 sec.	Constant speed	15 sec.
Dist. during acc.	62.5 ft.	Dist. during ret.	62.5 ft.	Dist. at const. speed	375 ft.
Angular velocity	6.25	Angular acc.	1.25	Angular ret.	1.25

Inertia

5,000
24,000
5,000
4,750
38,750

Drums, gears, sheaves 12,500
Motor $9280 + 32 \times 8.4^2 = 20,500$
Load $38,750 \div 32 \times 4^2 = 19,375$
Total inertia 52,375

Angular acc. moment $52,375 \times 1.25 = 65,500$ lb-ft
Angular ret. moment $52,375 \times 1.25 = 65,500$ lb-ft.
Friction assumed at 7.5% of 38,750 = 2,900 lb
Moment due to friction $2900 \times 4 = 11,600$

Load Diagram:

		Moment Lb.-ft. × Ang.Vol ÷ 550 = Horsepower	
1	(5000 + 1825) × 4 + 11,600 + 65,500	= 104,400	= 1186
2	(5000 + 1370) × 4 + 11,600 + 65,500	= 102,580	= 1166
3	(5000 + 1370) × 4 + 11,600	= 37,080	= 421
4	(5000 − 1370) × 4 + 11,600	= 26,120	= 297
5	(5000 − 1370) × 4 + 11,600 − 65,500	= −39,380	= −446
6	(5000 − 1825) × 4 + 11,600 − 65,500	= −41,200	= −469

Capacity of Hoist Motor

6,915,000
1,988,000
1,050,000
26⅔ | 9,953,000
373,000

$$\sqrt{\frac{\frac{1186^2+1166^2}{2}\times 5+\frac{421^2+297^2}{2}\times 15+\frac{446^2+469^2}{2}\times 5}{25+\frac{5}{3}}} = 610 \text{ hp}$$

Input with Rheostatic Control

Acceleration	(1186 + 1166) ÷ 2 × 5 =	5,880	hp.-sec.
Constant speed	(421 + 297) ÷ 2 × 15 =	5,390	hp.-sec.
Retardation	(446 + 469) ÷ 2 × 5 =	2,290	hp.-sec.
Total input to hoist		13,560	hp.-sec.
Average eff. of motor 88 per cent.	Input to motor	15,400	hp.-sec.
Input to motor 3.19 kw.-hr.	Kw.-hr. per ton	1.276	

Shaft, hp.-sec. $5000 \times 500 \div 550 = 4550$

Overall efficiency $= \frac{\text{Shaft hp.-sec.}}{\text{Total hp.-sec.}} = \frac{4,550}{15,400} = 29.5$ per cent.

Fig. 1.—Hoist calculations, No. 1.

Drum Calculations

A plain cylindrical drum has been assumed for simplicity; the question of conical and cylindro-conical drums will not be considered in this paper. The method of figuring special-shape drums is practically the same as for cylindrical drums, but somewhat more complicated as it is necessary to take into account the varying radius of the ropes on the drums and the varying rates of acceleration and retardation. Conical or cylindro-conical drums, in general, cut down the peak load during acceleration and retardation, but in many cases they introduce undesirable elements. Often the inertia of the special drums is high, which in a way defeats the purpose for which the drum is designed. In many cases considerable power is required to carry the cage into the dumping position, while with the cylindrical drum the cage will drift into the dumping position without the use of power.

In general, the drums should be as small in diameter as good rope practice dictates, for the smaller the drum diameter the greater will be the speed, which will permit of a fairly high motor speed or a lower gear reduction. Wherever possible the drum size and motor speed should be so selected that single reduction gears can be used where gearing is necessary. It is considered good practice to use not less than a

$1\frac{1}{8}$ in. rope on a 6-ft. diameter drum
$1\frac{1}{4}$ in. rope on a 7-ft. diameter drum
$1\frac{3}{8}$ in. on a rope 8-ft. diameter drum
$1\frac{1}{2}$ in. rope on a 9-ft. diameter drum
$1\frac{5}{8}$ in. rope on a 10-ft. diameter drum
$1\frac{3}{4}$ in. rope on a 12-ft. diameter drum

From the maximum capacity in 7 hr. the trips per hour can readily be determined. The time for caging varies from 4 to 10 sec., and should be kept as low as possible in order to cut down the rope speed and the capacity of the equipment. After the caging time has been selected, the running time can readily be determined.

Maximum Rope Speed

With a plain cylindrical drum, the maximum rope speed, in feet per second, is obtained by dividing the depth, in feet, by the running time per trip in seconds, minus one-half of the sum of the accelerating and retarding time. After the rope speed is obtained the maximum drum speed and angular velocity can readily be determined. The inertia of the drum should be obtained from the hoist builder, when possible. For estimating purposes, the inertia of a cylindrical drum with head

sheaves and gears can be closely approximated by allowing 200 lb. per sq. ft. of drum surface and assuming the radius of gyration at 3 in. inside of the drum surface.

The selection of the accelerating and retarding time depends on the hoisting conditions; this time will vary from 4 to 8 sec. for fast coal hoists and 10 to 20 sec. for slopes and deep metal mines. These values should be made as small as is practical, for the smaller they are the lower will be the maximum rope speed. The retarding time is frequently made less than the accelerating time, as the friction of the hoist assists during retardation. On the forms, a space is left for a sketch showing the size and shape of the drums. This sketch is particularly necessary when a special shape of drum is used in order that the turns and varying radius at different parts of the cycle may be shown. The item, Total Revolutions per Trip, is used only for special-shape drums.

Time

The distance traveled by the cages or skips can be readily determined from the rope speed and time. The angular acceleration and retardation is obtained by dividing the angular velocity by the accelerating and retarding time.

Inertia

With high-speed hoisting, inertia plays a prominent part and in many cases the power required during acceleration and retardation is the dominant factor in determining the capacity of the electrical equipment. Under the heading Inertia, the inertia of the various parts is determined based on the drum radius. The moment of inertia is the weight multiplied by the radius of the gyration squared and divided by 32. Where the motor is geared, the moment of inertia of the armature must be multiplied by the square of the gear ratio to refer it to the drum radius. A motor is selected that will probably be of sufficient capacity so that the inertia of the rotor can be estimated. If the weight of the rotor is known, the inertia can be calculated by assuming the radius of gyration to be 75 to 80 per cent. of the radius of the rotor, depending on the details of the construction. In some cases, it is necessary to go back after the load cycle has been calculated and select another size of motor.

The load part of the inertia consists of the weight of the material to be hoisted, the weight of the cages, cars, and the rope on both drums. This total weight is suspended from the drum rim so that the radius of gyration is the same as the drum radius. The addition of the individual weights is shown on the left-hand margin of sheet No. 1. After the total inertia of all moving parts is obtained, the moment due to angular acceleration and retardation can be readily calculated by multiplying the total inertia by the angular acceleration and retardation.

Friction of Hoist Parts

The friction of the hoist parts, including windage and friction of the cages in the shaft, has been the subject of much controversy. This friction is sometimes assumed on the basis of a certain hoist efficiency of 80 to 85 per cent. This method may result in considerable error during the high-speed portion of the cycle, where the actual load on the hoist motor is very low and when the friction load is really at its maximum. A load cycle for a deep mine hoist may show that during the constant high-speed portion of the cycle, the load on a 2000-hp. motor may go as low as 100 hp. or less. With a 15 per cent. loss, the friction load would figure 15 hp., while the actual friction load would probably be over 200 hp. The friction of a hoist based on the pull produced at the drum rim is practically constant throughout the cycle, so that a method involving the equivalent pull at the drum rim would seem to be much more logical. It has been the practice of the Westinghouse engineers to assume that the total friction of a hoist is based on a rope pull of 7½ per cent. of the total suspended weight on the drums for single-reduction geared hoists, and 5 and 5½ per cent. for direct-connected hoists. This rope pull, when multiplied by the radius of the drum, will give the moment at the drum rim due to friction. This value is assumed to represent the friction throughout the entire cycle. The value for frictional resistance can, however, be varied to quite an extent either way without vitally affecting the capacity of the equipment.

Load Diagram

With the foregoing information, which has involved calculations of the simplest nature only, the load at various points in the hoisting cycle can be determined, as outlined under the heading Load Diagram. The load diagram is determined by calculating the torque, or moment, in foot-pounds at the drum rim at the start, at the end of accelerating period, at the beginning of the constant-speed period, at the end of the constant-speed period, at the beginning of the retardation period, and at the end of the retardation period. This involves six equations for the plain cylindrical drum. Special drums frequently require a larger number of equations, which of course can be readily taken care of in the hoist calculation form.

The total moment at any point of the cycle will consist of the moment due to the load to be hoisted plus the unbalanced cable. This is obtained by multiplying the weight of the load and unbalanced cable by the drum radius; to this is added the friction moment and the accelerating and retarding moment if the hoist is accelerating or slowing down. The retarding moment has, of course, a negative value. The total moment must be supplied by the hoist motor. The cages and empty cars balance each

other and affect the load cycle only during acceleration and retardation. As the loaded cage ascends and the empty cage descends, the amount of unbalanced cable changes rapidly. At the end of the cycle, the suspended cable is on the empty side and its weight tends to balance the load. Where very deep mines are involved, the weight of the suspended cable will often more than balance the load, causing the power to become negative.

The angular velocity is the linear speed, in feet per second, at the radius of 1 ft. (0.3 m.). As the total moment is the equivalent rope pull at 1 ft. radius, the product of the two will give the rate of work, in foot-pounds per second. If this value is divided by 550, the result will be horsepower. It is, therefore, only necessary to multiply the total moment by the angular velocity and divide by 550 to obtain the horsepower values.

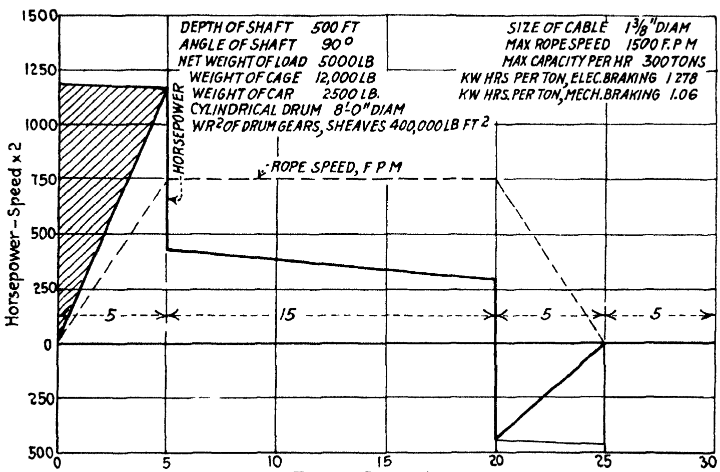


FIG. 2.—CURVE NO. 1; HOISTING CYCLE 2100 TONS PER DAY.

Having obtained the entire load cycles in six equations, the results can be plotted, as shown in Fig. 2. It will be readily noted that a considerable portion of the motor capacity is due to acceleration and retardation. This curve indicates clearly the desirability of keeping all moving weights down to as low a value as possible, consistent with strength, reliability, and good practice.

Capacity of Hoist Motor

The capacity of an electric motor depends on the temperature to which the various parts rise above the surrounding atmosphere. The temperature rise depends on the motor losses, most of which are dissipated as heat. If these losses were proportional to the load upon the

motor, it would be a simple matter to determine the average load and select a motor of the same capacity. In the foregoing case, the average load upon the motor is 450 hp. The losses, however, vary closely as the square of the load, so that to obtain a rating that will give an equivalent heating, it is necessary to determine the rating by taking into account the losses. The prevalent method for obtaining this equivalent rating is to calculate the root mean square rating, as shown under the heading Capacity of Hoist Motor. Each part of the cycle is taken separately and the equivalent heating calculated by adding the squares of the two horsepower values, dividing the sum by 2, and multiplying the result by the time. Where there is a fairly large difference between the horsepower values in one part of the cycle, a more accurate result is obtained by adding to the sum of the squares the product of the two values and dividing the result by 3 before multiplying by the time. This is considered later in determining the capacity of the synchronous motor, which is part of a motor-generator set.

The final value obtained is 610 hp., showing that a 600-hp. motor is required and also indicating that the average value in this case must be increased by 33⅓ per cent. to obtain the correct value. The capacity obtained is based on reversing the motor to obtain retardation. If mechanical brakes are used, the last part of the calculation will be left out and the capacity required will be 579 hp. instead of 610. The same motor would be recommended, however.

There has been considerable argument regarding time used in a calculation for capacity. A fairly high-speed motor will dissipate heat at full speed much more rapidly than when standing idle. For this reason, the time will have a different real value at different parts of the cycle. Some engineers take the full time at full speed, a certain percentage of the accelerating and retarding time, and a smaller percentage of the rest period. The writer has adopted the scheme of using the full time during actual operation, one-third of the time during the rest period for geared motors, and one-half of the rest period for large direct-connected motors. The difference between the ability to dissipate heat at full speed and at a standstill is greater for a high-speed motor than for a low-speed. When forced ventilation is used on the hoist motor, full time can be used during the entire cycle.

From the heating standpoint, a 600-hp. motor will have ample capacity. The motor must, however, be checked for maximum load. Most induction motors are designed to have a pull-out torque of twice the full-load torque. In this case, if the motor has a pull-out torque of 2, the power required during the accelerating period will be very close to the pull-out torque of the motor, so that there will be no margin for a drop in line voltage, which is apt to occur even on the best-regulated power lines. As the pull-out torque of an induction motor varies directly as the

square of the impressed voltage, a small drop in line voltage will cause a much larger reduction in the pull-out torque of the motor. It is recommended that the pull-out of an induction motor be selected at a value 25 to 30 per cent. greater than the maximum value on the load cycle to take care of a possible drop of at least 10 per cent. in line voltage. In the present case, this condition will be met by providing a motor with a pull-out torque of 2.5.

Speed of Motor

The speed of the motor depends on several factors. A high-speed motor will be cheaper and have a better performance, with regard to efficiency and power factor, than a low-speed motor. The high-speed motor, on the other hand, requires a high gear ratio and the equivalent inertia at the drum rim is very high. The low-speed motor is larger and more costly and has a poorer performance than the high-speed. the inertia will be greater than the higher speed motor, but in most cases the lower gear ratio will more than make up for this difference. In general, a motor speed should be selected that will permit of single reduction gears. Herringbone gears can be obtained with ratios as high as 15 to 1. The final speed of the motor is generally a compromise to obtain the best all-around conditions and will be influenced, to some extent, by the available speed of motors as developed by the various manufacturers. In general, motors up to 200 hp. will range from ten to fourteen poles; from 200 to 600 hp., fourteen to sixteen poles; and from 600 hp. up, sixteen to twenty poles.

Input with Rheostatic Control

From the hoisting cycle, the kilowatt-hours per ton and the overall efficiency of the hoist can be estimated, as shown under the heading Input with Rheostatic Control. The output of the hoist motor, in horsepower-seconds, is calculated from each part of the cycle and an average efficiency of the hoist motor is assumed at 88 per cent. The total input, in horsepower-seconds, can be readily converted into kilowatt-hours from which the value 1.275 kw.-hr. per ton is obtained.

The shaft horsepower-seconds for the actual work done in hoisting the coal is obtained by mutiplying the load hoisted by the distance and dividing by 550. This value divided by the input to the alternating-current hoist motor will give the overall efficiency of the cycle, which in the present case is 29.5 per cent. If mechanical brakes are used, which do not require power from the line, the overall efficiency will be 35.5 per cent. Where considerable power is required during the retardation portion of the cycle, it is not possible to use mechanical brakes unless they are of a design that can readily take care of the amount of power involved. In many cases it is necessary to plug the motor to obtain the

proper rate of retardation. This is necessary where the mechanical brakes do not have sufficient capacity to absorb the power required during the braking period.

The complete load cycle is plotted in Fig. 2, which also shows the rope speed, kilowatt-hours per ton, and other information in regard to the hoist. A glance at the load cycle will show why the efficiency is low. The shaded part of the accelerating portion of the cycle indicates the power loss in the rheostat; this loss is unavoidable when rheostatic control is used. If electric braking is used, the entire area of the quadrilateral representing the retardation period will be lost in the rheostat. The use of conical or cylindro-conical drums will, in some cases, reduce the capacity of the motor required, and also reduce the rheostatic losses by reducing the load during the accelerating and retarding periods.

Control of Hoist Motor

The control for a hoist motor is particularly important, since with improper control the success of the entire installation is jeopardized. For small low-voltage hoists, a drum controller can be recommended in most cases for either alternating or direct current. For the medium and larger sizes of a.c. hoists, 2200-volt motors are becoming standard and the most satisfactory control is the magnetic type with air-brake switches in both primary and secondary circuits. The high-voltage air-brake magnetic contactor is to be recommended for the primary, especially where frequent operations are necessary. The magnetic type of control readily lends itself to the application of safety devices, such as automatic acceleration, overload trip, low-voltage trip, brake magnets, interlocking reverse switches, geared and hatchway limit switches, etc.

Slope hoists should be equipped with at least one more accelerating point than vertical hoists of the same capacity, with a lower torque provided on the first notch to permit of slow starts, which is particularly necessary where slack cable exists.

The liquid rheostat is used to some extent in connection with a.c. hoists. This is a simple piece of apparatus and will give satisfactory service where a supply of good cooling water is available. This type of rheostat makes the best showing with the larger a.c. hoists and where the cycle is not too rapid. For most installations, however, the magnetic type of controller is to be preferred.

Where the maximum rope speed exceeds 1800 ft. per min. (548 m.) and the cycles approach 3 per min., the application of an a.c. motor to a hoist becomes questionable. For such high rope speeds and rapid cycles, the inertia effect of the a.c. motor is great, the power loss during acceleration and retardation high, the peak loads upon the power system in some cases prohibitive, and the braking system questionable. Under such conditions the best solution seems to be the application of a d.c.

hoist motor receiving power from either a flywheel or synchronous motor-generator set. The inertia of a d.c. motor is, in most cases, much lower than an a.c. motor; and the control, where field control is utilized, is vastly superior, as rapid changes can be made in operating conditions and electric braking can be safely and economically obtained entirely independent of the power supply.

Alternating-current Hoist Motor Calculations for an Output of 3000 *Tons*

Let us assume the same set of specifications, except that the output per day is 3000 tons. The calculations for an a.c. motor are given in hoist calculation sheet No. 2, Fig. 3, which shows that the cycle must be made in 21 sec. instead of 30 sec. The maximum rope speed has been increased from 1500 ft. per min. (457 m.) to 2730 ft. per min. (832 m.) and the drum speed to 108.5 r.p.m.

An a.c. motor is selected that has a 40° rating of 1500 hp. or a 50° rating of 1875 hp. wound for 2200 volts, three phase, 60 cycles, twenty pole. The WR^2 of the rotor of this motor is over five times that of the 600-hp. motor. The gear ratio, however, is only 3.22 to 1, so that the inertia at the drum rim, in pounds-feet squared, is really less than for the 600-hp. motor. In going through the complete set of calculations it is found that the capacity of the motor figures out 1870 hp. and the pull out required will have to be at least 2, based on the 50° rating.

It is possible that the margin in rating is not sufficient if a large amount of rock is to be handled. The next size of motor is selected having a capacity of 2000 hp. at 40° rise. The inertia of the rotor of this motor, as shown in hoist calculation sheet No. 3, Fig. 4, is nearly two times that of the 1500-hp. motor. The load diagram shows a large increase in the power required during acceleration and retardation. This increase is very apparent in figuring out the capacity of the motor, which proves to be 2340 hp. A 2000-hp., 40° motor will probably have a 50° rating of about 2350 hp., so that no advantage is obtained by adopting the larger motor. Applying a large motor on a high-speed cycle produces a pyramiding effect, for the added capacity is absorbed by the increased power required by the added inertia of the rotor.

The load cycles for both alternating-current motors are plotted in Fig. 5, which graphically shows the comparison. The efficiency is particularly low, especially with a larger motor, being 14.5 per cent. for the 1500-hp. and 11.8 per cent. for the 2000-hp. motor. In either case the power required during the retardation period is too great to absorb by mechanical brakes.

If the hoisting conditions do not permit the installation of any but an a.c. motor, it is sometimes possible to meet such a high-speed cycle by applying two motors instead of one. By using the same speed or possibly a somewhat higher speed motor, the total inertia may be considerably

Customer	A. I. M. E.			Neg.	000	Date	7-1-21
Power	2200	Volts	3	Ph.	60	Cy.	

General Data

Depth of shaft	500 ft.	Total rev. per trip	
Angle of slope	90°	Max. rope speed	$\frac{500}{16-5}$ = 45.5 ft. per sec. = 2730 ft. per min.
Net weight of load	5000 lb.	Max. drum speed	108.5 r.p m. = 1.81 r p.s.
Weight of cage, skip	12,000 lb.	Size of rope	1⅝-in. dia. wt. = 3.65 lb. per ft.
Weight of car	2500 lb.	Size and shape of drums	8 ft 0 in. dia Cylindrical
Balanced, unbalanced	Bal.	WR^2 drums, gears, sheaves	400,000 lb.-ft.^{2}
Max capacity per day	3000 tons	Angular velocity	$2\pi \times 1\ 81 = 11\ 38$
Trips per hour	171		
Time for caging	5 sec.		
Running time per trip	16 sec.		
Total time per trip	21 sec.		

Time

Acceleration	5 sec.	Retardation	5 sec.	Constant speed	6 sec.
Dist. during acc.	113 8 ft.	Dist during ret	113 8 ft.	Dist. at const. speed	272 4 ft.
Angular velocity	11 38	Angular acc	2.275	Angular ret.	2 275

Inertia

Drums, gears, sheaves	12,500	Angular acc. moment	47,475 × 2.275 = 108,000
Motor 48,500 ÷ 32 × 3 22^2 =	15,600	Angular ret. moment	47,475 × 2.275 = 108,000
Load 38,750 − 32 × 4^2 =	19,375	Friction assumed at	7½ % of 38,750 = 2900 lb.
Total inertia	47,475	Moment due to friction	2900×4 = 11,600

Load Diagram

		Moment Lb -ft.	Horsepower
1	(5000 + 1825) × 4 + 11,600 + 108,000 =	146,900	3030
2	(5000 + 995) × 4 + 11,600 + 108,000 =	143,580	2965
3	(5000 + 995) × 4 + 11,600 =	35,580	735
4	(5000 − 995) × 4 + 11,600 =	27,620	572
5	(5000 − 995) × 4 + 11,600 − 108,000 =	−80,380	−1660
6	(5000 − 1825) × 4 + 11,600 − 108,000 =	−83,700	−1730

Capacity of Hoist Motor

44,800,000
2,595,000
14,320,000
17⅔ | 61,715,000
3,495,000

$$\sqrt{\frac{\frac{3030^2 + 2965^2}{2} \times 5 + \frac{735^2 + 572^2}{2} \times 6 + \frac{660^2 + 1730^2}{2} \times 5}{16 + \frac{5}{3}}} = 1870 \text{ hp.}$$

Input with Rheostatic Control

Acceleration	(3030 + 2965) − 2 × 5 =	15,000	hp -sec.
Constant speed	(735 + 572) ÷ 2 × 6 =	3921	hp.-sec.
Retardation	(1660 + 1730) ÷ 2 × 5 =	8475	hp. sec.
Total input to hoist		27,396	hp.-sec.
Average eff. of motor	88 per cent Input to motor	31,100	hp. sec.
Input to motor	6 5 kw -hr. Kw.-hr. per ton	2.6	
Shaft hp -sec.	5000 × 500 ÷ 550 = 4550		

$$\text{Overall efficiency} = \frac{\text{Shaft hp -sec.}}{\text{Total hp.-sec}} = \frac{4550}{31100} = 14.6\ \%.$$

Fig. 3.—Hoist calculations, No. 2.

Customer	A. I. M E.			Neg.	000	Date	7-1-21
Power	2200	Volts	3	Ph.	60	Cy.	

General Data

Depth of shaft	2500 ft.	Total rev. per trip	
Angle of slope	90°	Max. rope speed	$\frac{500}{16-5} = 45.5$ ft. per sec. = 2730 ft. per min.
Net weight of load	5000 lb.	Max. drum speed	108 5 r p.m. = 1.81 r.p.s.
Weight of cage, skip	12,000 lb	Size of rope	1⅜-in. dia. wt. = 3.65 lb. per ft.
Weight of car	2500 lb.	Size and shape of drums	8 ft 0 in. dia. Cylindrical
Balanced, unbalanced	Bal.	WR^2 drums, gears, sheaves	400,000 lb.-ft.^{2}
Max. capacity per day	3000 tons	Angular velocity	$2\pi \times 1.81 = 11.38$
Trips per hour	171		
Time for caging	5 sec.		
Running time per trip	16 sec		
Total time per trip	21 sec.		

Time

Acceleration	5 sec.	Retardation	5 sec.	Constant speed	6 sec.
Dist. during acc.	113 8 ft.	Dist. during ret.	113 8 ft	Dist. at const. speed	272.4 ft
Angular velocity	11.38 ft.	Angular acc.	2 275	Angular ret.	2 275

Inertia

Drums, gears, sheaves	12,500 lb.-ft ^{2}	Angular acc. moment	$61{,}375 \times 2.275 = 139{,}500.$
Motor	$91{,}000 \div 32 \times 3\ 22^2 = 29{,}500$ lb -ft.	Angular ret. moment	$61{,}375 \times 2.275 = 139{,}500.$
Load	$38{,}750 \div 32 \times 4^2 = 19{,}375$ lb -ft	Friction assumed at	7½ % of 38,750 = 2,900 lb
Total inertia	61,375 lb -ft	Moment due to friction	$2900 \times 4 = 11{,}600$

Load Diagram

		Moment Lb.-ft.	Horsepower
1	$(5000 + 1825) \times 4 + 11{,}600 + 139{,}500 =$	178,400	3690
2	$(5000 + 995) \times 4 + 11{,}600 + 139{,}500 =$	175,080	3620
3	$(5000 + 995) \times 4 + 11{,}600 =$	35,580	735
4	$(5000 - 995) \times 4 + 11{,}600 =$	27,620	572
5	$(5000 - 995) \times 4 + 11{,}600 - 139{,}500 =$	−111,880	−2310
6	$(5000 - 1825) \times 4 + 11{,}600 - 139{,}500 =$	−115,200	−2380

Capacity of Hoist Motor

$$\begin{array}{r} 66{,}500{,}000 \\ 2{,}595{,}000 \\ \underline{27{,}500{,}000} \\ 17\tfrac{2}{3}\,|\,\underline{96{,}595{,}000} \\ 5{,}470{,}000 \end{array} \qquad \sqrt{\frac{\frac{3690^2 + 3620^2}{2} \times 5 + \frac{735^2 + 572^2}{2} \times 6 + \frac{2310^2 + 238^2}{2} \times 5}{16 + \frac{5}{3}}} = 2340 \text{ hp.}$$

Input with Rheostatic Control

Acceleration	$(3690 + 3620) \times 5 \div 2 =$	18,250	hp.-sec.
Constant speed	$(735 + 572) \times 6 \div 2 =$	3,921	hp.-sec.
Retardation	$(2310 + 2380) \times 5 \div 2 =$	11,725	hp.-sec.
Total input to hoist		33,896	hp.-sec.
Average eff. of motor	88 per cent. Input to motor	38,500	hp.-sec.
Input to motor	7 98 kw.-hr. Kw.-hr. per ton	3 19	
Shaft hp.-sec	$5000 \times 500 \div 550 = 4550$		

$$\text{Overall efficiency} = \frac{\text{Shaft hp.-sec.}}{\text{Total hp.-sec.}} = \frac{4{,}550}{38{,}500} = 11.8 \text{ per cent.}$$

Fig. 4.—Hoist calculations, No. 3.

less by the use of two motors. This depends somewhat on the motors that have been developed and built. It would, of course, be possible to develop a special motor having very much less inertia than either of the motors shown in Figs. 3 and 4. The development of this motor, however, would probably cost several thousands of dollars, which would make it non-competitive. The control of two motors will be a little more com-

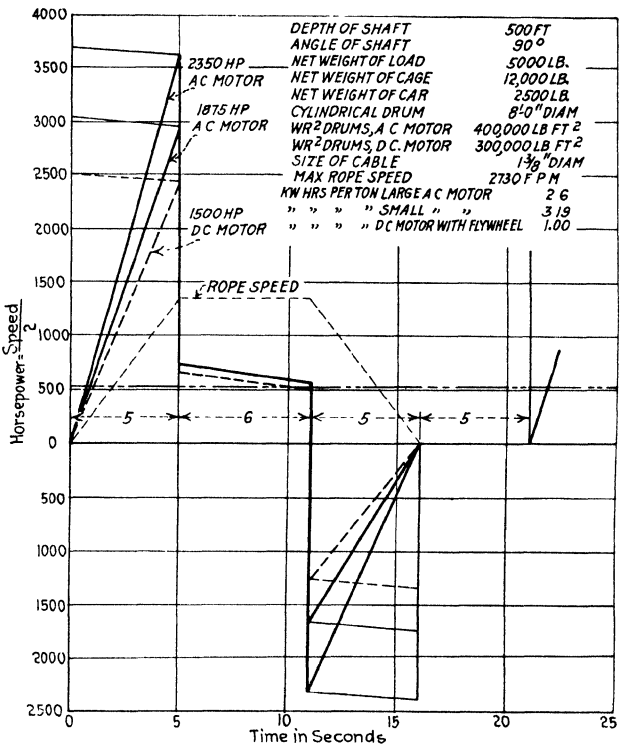


FIG. 5.—CURVE No. 2; 3000 TONS PER DAY, A.C. AND D.C. HOISTING SYSTEMS.

plicated than the control for one, but the cost of the entire installation may not be any greater than when using one large motor of a capacity greater than the combined capacity of two motors.

The theoretical power required to drive the hoist is represented, in Figs. 2 and 5, by the heavy diagonal lines starting at zero. The actual power taken by the hoist motor when an alternating-current motor is used is represented by the light line starting at a high value. In other words, practically all of the high peak load at the moment of starting is

absorbed in the rheostat. The same applies to the retardation portion of the cycle, except that when electric braking is used with an a.c. motor all of the power represented by the quadrilateral is drawn from the line and wasted in the rheostat.

Calculations for a Direct-current Hoist Motor

Hoist calculation sheet No. 4 shows the same problem except that a direct-connected hoist motor is used instead of a geared alternating-current hoist motor. The weight and inertia of the d.c. armature is of course much greater than any of the rotors of the a.c. motors mentioned, but being direct connected the equivalent inertia at the drum is much less. There being no gears, the inertia of the drums and sheaves is considerably less. The friction is also somewhat less, being figured at 5½ per cent. instead of 7½ per cent. of the total suspended weight.

The hoisting cycle shows much smaller accelerating and retarding values and the capacity of the motor is but 1485 hp. The load cycle is plotted as a heavy dotted line in Fig. 5. With field control, there are no rheostatic losses in the main armature circuits, so that the power follows the diagonal line from the origin instead of the vertical line, as in the case of the a.c. motor. The light dotted line starting at 2500 hp. indicates the torque that must be delivered by the hoist motor. In other words, the hoist motor delivers a high torque at the start with a low expenditure of power, for there are no rheostatic losses. As the current in the hoist motor is proportional to the torque, this line also represents the current from which the heating can be calculated. The power represented in the upper triangle during the accelerating period is only apparent and not real when using a field control system. The power represented in the upper triangle of the retarding part of the cycle is returned to the system instead of being drawn from the line, as with the a.c. motor.

All power taken from the power system, when using a flywheel motor-generator set, passes through the induction motor and since the flywheel cannot change its speed quickly, the load on the induction motor cannot change quickly. As the load comes on the hoist motor, it is transferred through the generator to the induction motor. As soon as the load on the induction motor reaches a predetermined value, a torque motor on the slip regulator causes a resistance to be inserted in the secondary of the induction motor. This causes the speed to drop and the flywheel to give up energy. By this means the maximum load on the induction motor is fixed at a predetermined value. At any time during the cycle that the load on the hoist motor is above this value, the flywheel will supply the surplus, while if the load is below this value, the flywheel will store up energy keeping practically a constant load on the induction motor.

Voltage control with Flywheel

When making calculations for direct-current field-control system, it is not necessary to fill in the spaces at the bottom of the calculation sheet under the heading Input with Rheostatic Control. With the d.c. field-control system, the d.c. hoist motor receives its power from a d.c. generator, which in turn is driven by an a.c. wound-rotor induction motor or a synchronous motor. The flywheel type of motor-generator set will be taken up first. Calculation sheet No. 5, Fig. 7, shows how the capacity of the generator, the weight of the flywheel, the capacity of the induction motor, and the overall efficiency are obtained.

At the top of the sheet are four columns, the items in the first, which is headed Hoist-motor Output, are obtained from the load cycle on calculation sheet No. 4, Fig. 6.

The second column gives the input to the hoist motor at the various points in the cycle. These values are obtained by adding the variable losses of the hoist motor to the output values in the first column. These losses consist of the copper, iron, friction, and windage. The excitation is considered a fixed loss to be taken care of later. The best way to obtain the variable losses is to plot a loss curve for the particular motor, as shown in Fig. 8. By means of this curve, the losses at each point in the cycle can be readily determined. During the retarding portion of the cycle, the hoist motor is acting as a generator to return power to the flywheel; consequently the losses must be subtracted instead of added.

The third and fourth columns are headed Generator Input, the third being the apparent input and the fourth the real. The third column is obtained by adding the variable losses of the generator to the values in the second column; that is, the copper and iron losses. The excitation, friction, and energy are fixed losses to be taken care of later. The generator variable loss curve is also plotted in Fig. 8.

The fourth column is the same as the third column, except for the first and last values, which represent torque and not real horsepower in the third column. The real values are substituted in the fourth column and consist of the copper losses of the hoist motor and generator. The iron loss, friction, and energy of the hoist motor and the iron loss of the generator are practically zero at the moment of starting and stopping.

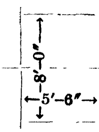
The fifth column, Horsepower-seconds Input, is used to obtain the average power input to the generator required during the entire cycle. The horsepower-seconds during each portion of the cycle is readily obtained by taking the average value and multiplying by the time. The power during retardation is subtracted, for it represents returned power. The total net value of 9137.5 hp.-sec. is divided by the total time of the cycle or 21 sec. The result is 435 hp. for the average power delivered to the generator.

The values in the fourth column are plotted in Fig. 9 and a horizontal line drawn at 435 hp. represents the average input. This curve shows that for complete equalization the power in the shaded portion above the

Customer	A. I. M. E.			Neg	000	Date	7-1-21
Power	2200	Volts	3	Ph.	60	Cy.	

General Data

Depth of shaft	500 ft.	Total rev. per trip	
Angle of slope	90°	Max. rope speed $\frac{500}{16-5}$ = 45.5 ft. per sec. = 2730 ft. per min.	
Net weight of load	5 000 lb	Max. drum speed	108.5 r.p.m. = 1.81 r.p.s.
Weight of cage	12,000 lb.	Size of rope	1⅛-in. dia. wt. = 3.65 lb. per ft.
Weight of car	2500 lb.	Size and shape of drums	8 ft. 0 in. dia. Cylindrical
Balanced, unbalanced	Bal.	WR^2 drums, sheaves	1800,000 lb.-ft.^{2}
Max. capacity per day	3000 tons	Angular velocity	$2\pi \times 1.81 = 11.38$
Trips per hour	171		
Time for caging	5 sec		
Running time per trip	16 sec.	Drum	
Total time per trip	21 sec.		



Time

Acceleration	5 sec.	Retardation	5 sec.	Constant speed	6 sec.
Dist. during acc	113.8 ft.	Dist. during ret.	113 8 ft.	Dist. at const. speed	272 4 ft.
Angular velocity	11.38	Angular acc.	2 275	Angular ret.	2.275

Inertia

Drums, sheaves	9400 lb.-ft.^{2}	Angular acc. moment	37,375 × 2.275 = 85,000
Motor	8600 lb.-ft.^{2}	Angular ret. moment	37,375 × 2.275 = 85,000
Load 38,750 ÷ 32 × 4^2 =	19,375 lb -ft^2	Friction assumed at 5½ per cent. of 38,750 =	2130 lb.
Total inertia	37,375 lb -ft.^{2}	Moment due to friction	2130 × 4 = 8520

Load Diagram

		Moment Lb.-ft.	Horsepower
1	(5000 + 1825) × 4 + 8520 + 85,000 =	120,820	2500
2	(5000 + 995) × 4 + 8520 + 85,000 =	117,500	2435
3	(5000 + 995) × 4 + 8520 =	32,500	673
4	(5000 − 995) × 4 + 8520 =	24,540	508
5	(5000 − 995) × 4 + 8520 − 85,000 =	−60,460	−1250
6	(5000 − 1825) × 4 + 8520 − 85,000 =	−63,800	−1320

Capacity of Hoist Motor

$$\begin{array}{r} 30,300,000 \\ 2,136,000 \\ 8,250,000 \\ \hline 18\tfrac{1}{2}\,|\,40,686,000 \\ 2,200,000 \end{array} \quad \sqrt{\frac{\frac{2500^2 + 2433^2}{2} \times 5 + \frac{673^2 + 508^2}{2} \times 6 + \frac{1250^2 + 1320^2}{2} \times 5}{16 + \frac{5}{2}}} = 1485 \text{ hp.}$$

Fig. 6.—Hoist calculations, No. 4.

435-hp. line must be supplied by the flywheel. The value of this power, in horsepower-seconds, is readily computed, as shown on calculation sheet No. 5, Fig. 7, which indicates that the total power to be absorbed is 7071 hp.-sec.

	Hoist-motor Output		Hoist-motor Input		Generator-input App.	Real		Hp.-sec. Input to Generator
1	2500 + 210	=	2710 + 170	=	2880	335		
2	2437 + 200	=	2637 + 163	=	2800	2800	+ 2 × 5 =	7837
3	673 + 36	=	709 + 32	=	741	741	+ 2 × 6 =	3921
4	508 + 30	=	538 + 28	=	566	566		11,758
5	−1250 + 70	=	−1180 + 50	=	−1130	−1130		
6	−1320 + 75	=	−1245 + 53	=	−1192	+ 83		12621
Total hp.-sec. input to generator								9137

Average input to generator $= \dfrac{\text{Total input}}{\text{Total time of cycle}} = \dfrac{9137}{21} = 435$ hp.

Size of Flywheel. See Curve No. 4.

Input to flywheel during acc. $\dfrac{2800 - 435}{2} \times 4.8 =$ 5670 hp.-sec.

Input to flywheel during const. speed $\dfrac{741 - 435 + 566 - 435}{2} \times 6 =$ 1311 hp.-sec.

Total input 6981 hp.-sec.

Normal speed of set 685 r.p.m. Normal speed at Rad. of Gyr. $= V =$ 254 ft. per sec.

Dia. of flywheel 10 ft. Speed at 10 per cent. Red. $= V_1 =$ 228.6 ft. per sec

Peripheral speed 21,600 ft per min.

Weight of flywheel $= \dfrac{\text{Hp-sec} \times 550 \times 64.4}{V^2 - V_1^2} = \dfrac{6981 \times 550 \times 64.4}{254^2 - 228.6^2} =$ 20,100 lb

Use 22,000 lb wheel Thickness 7 in

Size of Driving Motor	Full-load Loss	No-load Loss	
Av. input to generator	435		hp.
Motor and generator excitation	25	10	hp.
Exciter loss	2	2	hp.
Windage and friction of generator	15	15	hp.
Flywheel loss	20	20	hp.
Slip regulator loss	35		hp.
Iron loss a.c. motor		10	hp.
Friction and windage a.c. motor		10	hp.
No load copper loss a.c. motor		1	hp.
Total output a.c. motor	532		hp
Total input a.c. motor $= \dfrac{\text{Output}}{\text{Eff.}} = \dfrac{532}{0.92} =$	578 hp	68	hp.

Input to a.c. motor, hp.-sec. $=$ 578 × 21 = 12,138 = 2.51 kw.-hr.

Kw.-hr. per ton 1 Shaft hp.-sec. $= \dfrac{5000 \times 500}{550} =$ 4550 kw.-hr.

Overall efficiency $= \dfrac{\text{Shaft hp.-sec}}{\text{Input hp.-sec.}} = \dfrac{4550}{12138} =$ 37.5 per cent.

Capacity of Generator

35,550,000
2,367,000
7,350,000
21|45,267,000
2,150,000

$$\sqrt{\dfrac{\dfrac{2710^2 + 2637^2}{2} \times 5 + \dfrac{709^2 + 538^2}{2} \times 6 + \dfrac{1180^2 + 1245^2}{2} \times 5}{21}} = 1470 \text{ hp.} = 1097 \text{ kw.}$$

For hoist motor, recommend 1500 hp. For generator, recommend 1100 kw.

For a.c. motor, recommend 600 hp. For flywheel, recommend 22000 lb.

Fig. 7.—Hoist calculations, No. 5.

The capacity of the generator will be close to 1000 kw.; an economic speed for this capacity is 720 r.p.m. synchronous when driven by a 60-cycle motor. The automatic slip regulator in the secondary of the induction motor has a normal slip of about 3 per cent. and the motor windings and leads about 2 per cent., making a total of 5 per cent. At full load, the speed of the motor-generator set with regulator plates closed will be about 685 r.p.m. Calculation sheet No. 5 contains blanks for normal speed of set, diameter of flywheel, etc. Flywheels for such service in America almost exclusively consist of steel plates riveted together making

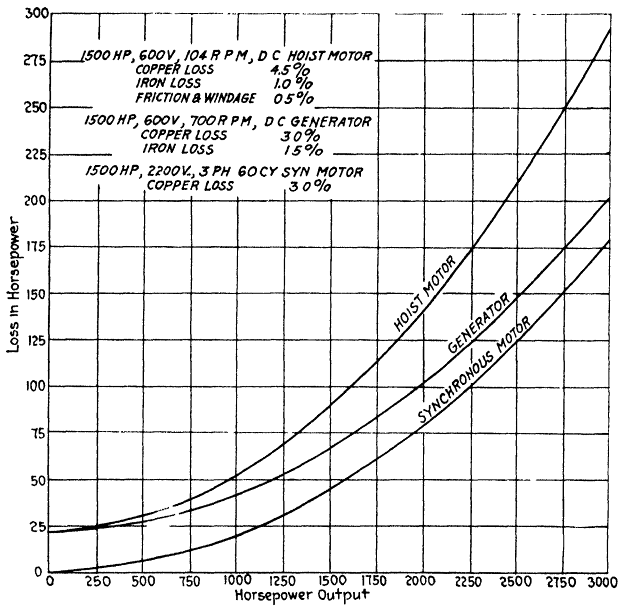


FIG. 8.—CURVE NO. 3; VARIABLE LOSSES.

a solid disk. Such a flywheel can be safely operated at 20,000 to 24,000 ft. per min. (6096 to 7312 m.) rim speed. The wheel selected will have a diameter of 10 ft. (3 m.) and a peripheral speed of 21,600 ft. per min. The radius of gyration of such a wheel is 0.707 times the radius to the rim, so that it is a simple matter to determine the speed, in feet per second, at the radius of gyration. The amount of speed reduction for flywheels in service of this kind varies from 10 to 20 per cent. The present problem will be worked out upon the basis of complete equalization with a slip of 10 per cent. In many cases complete equalization is not necessary and considerable saving in flywheel weight and losses can be obtained by using partial equalization. Partial equalization removes the heavy

peaks from the power system, but does not give a constant load upon the alternating-current motor.

The formula to obtain flywheel weights is derived from the simple laws of mechanics in which the energy contained in a moving mass is one-half MV^2. The difference between the energy contained in the wheel at full speed and reduced speed divided by 550 will represent the horsepower-seconds given up by the wheel when the speed is reduced The theoretical weight is estimated at 20,400 lb. (9253 kg.) and to use a margin of safety a 22,000-lb. (9979 kg.) wheel is recommended. This wheel will have a thickness of 7 in. (11.8 cm.) and will be surrounded by a sheet metal case to cut down the windage and prevent accidental contact by persons in the vicinity.

Capacity of Alternating-current Induction Motor

The average input to the generator was determined to be 435 hp. The various fixed losses not allowed for are listed under the heading Size of Driving Motor, on calculation sheet No. 5. The excitation of the hoist motor and generator, although not constant throughout the cycle, is assumed at an average value of 25 hp. The latest type of field control is quite a modification over the original Ward-Leonard system of control, in that the hoist-motor field is not kept at a fixed value but is overexcited during acceleration and retardation, operated at its normal value during constant speed, and underexcited during the rest period. By overexciting during acceleration and retardation, the armature current is kept at a lower value, which reduces the heating effect in the main circuit. This is of considerable importance in cycles where most of the heating occurs during the accelerating and retarding period. The exciting current is reduced during the rest period to cut down the field heating and incidentally to save a small amount of power; this reduction is generally about 30 per cent., which reduces the heating 50 per cent. Further reduction is liable to slow up operations, due to the time element of the hoist-motor field. In addition, higher speeds are sometimes obtained by several notches of field weakening of the hoist-motor fields. This scheme is, however, more prevalent in reversing rolling-mill practice than in hoisting.

The generator field varies with the speed and, by means of a comparatively new type of relay, the acceleration and retardation is entirely automatic, during which time the main armature current is held at a predetermined fixed value. The relays are so sensitive that the usual variations in current due to notching are entirely absent. The absence of variation in current during acceleration and retardation should be of considerable benefit in keeping down rope wear. By means of geared limit switches, the retardation can be made entirely automatic and independent of anything the operator may do.

The next fixed loss is that of the exciter itself, which is estimated at 2 hp. The windage and friction of the generator are fixed values, estimated at 15 hp. The flywheel loss is rather difficult to estimate and test results on existing wheels are rather erratic; the value of 20 hp. should be close to the actual power loss. The loss in the slip regulator varies throughout the cycle, the average is estimated to be approximately 7 per cent., or about 35 hp.

The total output of the a.c. motor will be 532 hp. A conservative value of 92 per cent. is selected for the efficiency so that the input from the power line will be 578 hp. This value is readily converted into horsepower-seconds and kilowatt-hours, resulting in a value of 2.52 kw.-hr. per cycle; this is equivalent to 1 kw.-hr. per ton of coal hoisted.

The overall efficiency is readily calculated by dividing the actual work done, in horsepower-seconds, by the input to the a.c. motor. The actual work done is designated as shaft horsepower-seconds and is the product of the weight of the coal by the depth divided by 550; the efficiency, as shown, is 37.5 per cent. Compared with the value obtained for the corresponding a.c. motor, there is not only a large difference in actual efficiency but also with the a.c. motor the momentary peak on the line is from 3400 to 4000 hp., depending on which a.c. motor is used, while the peak on the d.c. system is but 578 hp., or a difference of 2800 to 3800 hp. Where power is charged for on a basis of momentary peaks, the penalty caused by the use of an a.c. motor is excessive.

Capacity of Direct-current Generator

The capacity of the d.c. generator is obtained, in a manner similar to that used to determine the capacity of the hoist motor, by using the values in the second column at the top of calculation sheet No. 5. The entire time is used in the denominator, for the generator operates at practically full speed during the entire cycle. The result gives a capacity slightly smaller than the hoist motor, because the time is taken at a different value. The capacity of the generator should be 1100 kw.

Determining the No-load Losses

It is frequently desirable to know the no-load loss of the motor-generator set or the power required to drive the set during the rest period. If the rest period is of short duration, the motor fields are kept excited; if the period is of longer duration, the motor fields are frequently opened which cuts down the no-load losses about 10 hp. The total no-load losses with motor fields excited are shown under the heading No-load Losses in calculation sheet No. 5. This value is 68 hp. with motor fields excited and about 58 hp. if the motor fields are open.

At the bottom of calculation sheet No. 5, recommendations for the capacity of the hoist motor, generator, a.c. motor, and weight of flywheel

are made. These recommendations are often influenced by the available machines that have been built and may be the cause of a variation in the capacities recommended by different manufacturers.

Where the power system is large and where momentary peak loads are not penalized, a synchronous motor can be used on the motor-generator set. The calculations necessary for such a system are shown on calculation sheet No. 6, Fig. 10. Calculation sheet No. 4 is used to determine the load cycle and capacity of the hoist motor.

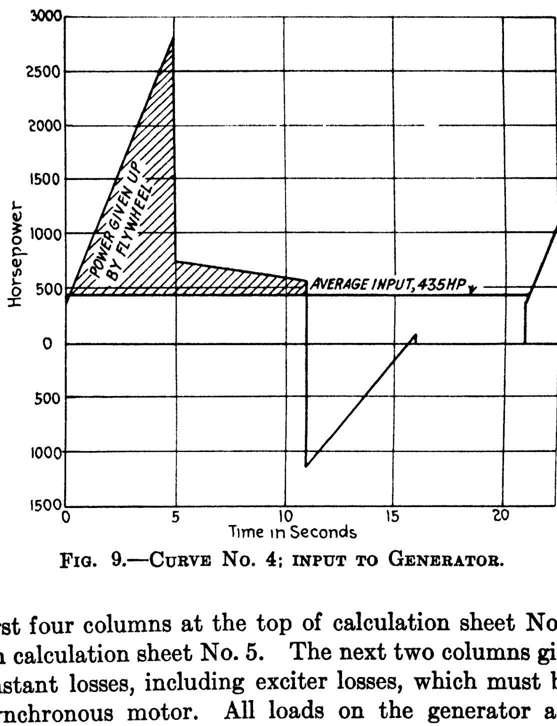


FIG. 9.—CURVE NO. 4; INPUT TO GENERATOR.

The first four columns at the top of calculation sheet No. 6 are the same as on calculation sheet No. 5. The next two columns give the generator constant losses, including exciter losses, which must be supplied by the synchronous motor. All loads on the generator are directly transmitted and must be supplied by the synchronous motor, while with the flywheel this was not directly the case, since the flywheel takes care of all peak loads and sudden fluctuations. The load on the synchronous motor follows the load on the hoist motor with the exception that all losses in the system are added to the load of the synchronous motor.

The first column in the second row gives the output of the synchronous motor at each point of the cycle; the next three columns give the internal losses of the synchronous motor, and these losses added to the output column give the values in the fifth column, which is the total input to the synchronous motor at each point in the cycle. The total

horsepower-seconds input is readily calculated as shown in the last column. Most of the retardation part of the cycle is negative while a part of it is positive; the net value, however, is readily computed.

	Hoist Motor Output	Hoist Motor Input	Generator App. Input	Generator Real Input	Total Excitation	Generator Fr. and Windage
1	2500	2710	2880	355	36	15
2	2437	2637	2800	2800	43	15
3	673	709	741	741	43	15
4	508	538	566	566	43	15
5	−1250	−1180	−1130	−1130	43	15
6	−1320	−1245	−1192	+ 83	36	15
7			Rest Period		28	15
8			Rest Period		28	15

	Synchronous-motor Output	Losses: Iron	Losses: Copper	Losses: Fr. and Wind.	Synchronous-motor Input		Input, Hp.-sec.
1	386	15	3	15	419	+ 2 × 5 =	8672
2	2858	15	162	15	3050		
3	799	15	12	15	841	÷ 2 × 6 =	4506
4	624	15	7	15	661		13178
5	−1072	15	23	15	− 1019		2122
6	+ 134	15	1	15	+ 165		11056
7	43	15	1	15	74		370
8	43	15	1	15	74		

Total input, hp.-sec. 11,426

Total kw.-hr. input 2.36 kw.-hr per ton 0 945

$$\text{Shaft hp.-sec.} = \frac{5000 \times 500}{550} = 4550$$

$$\text{Overall efficiency} = \frac{\text{Shaft hp -sec.}}{\text{Total input hp -sec.}} = \frac{4550}{11,426} = 39\ 8 \text{ per cent}$$

Capacity of Generator

35,550,000
2,367,000
7,350,000
21|45,267,000
2,150,000

$$\sqrt{\frac{\frac{2710^2+2637^2}{2}\times 5+\frac{709^2+538^2}{2}\times 6+\frac{1180^2+1245^2}{2}\times 5}{21}} = 1470 \text{ hp.} = 1097 \text{ kw.}$$

Capacity of Synchronous Motor

15,665,000
2,400,000
1,700,000
21|9,765,000
940,000

$$\sqrt{\frac{\frac{(386^2+2058^2)+386\times 2858}{3}\times 5 + 799^2 + 624^2 \times 6 + \frac{1072^2}{3}\times 4.45}{21}} = 970 \text{ hp.}$$

For hoist motor recommend 1500 hp., 600 volt, 108.5 r.p m., d c. motor

For generator recommend 1100 kw , 600 volt, 720 r p.m., d.c. generator

For synchronous motor recommend 1500 hp., 2200 volt, three-phase, 60 cycle, 720 r.p m. syn motor

Fig. 10.—Hoist calculations, No. 6.

From the total horsepower-seconds, the input is found to be 2.31 kw.-hr. and 0.925 kw.-hr. per ton. The overall efficiency is found to be 40.8 per cent. or a little over 3 per cent. better than when using the fly-

wheel set; the difference is largely due to the flywheel and slip regulator loss.

The no-load loss of the set is 74 hp., or somewhat larger than with the flywheel set; as the synchronous motor is much larger than the induction motor, its losses, including excitation, more than make up for the flywheel loss with the induction motor set.

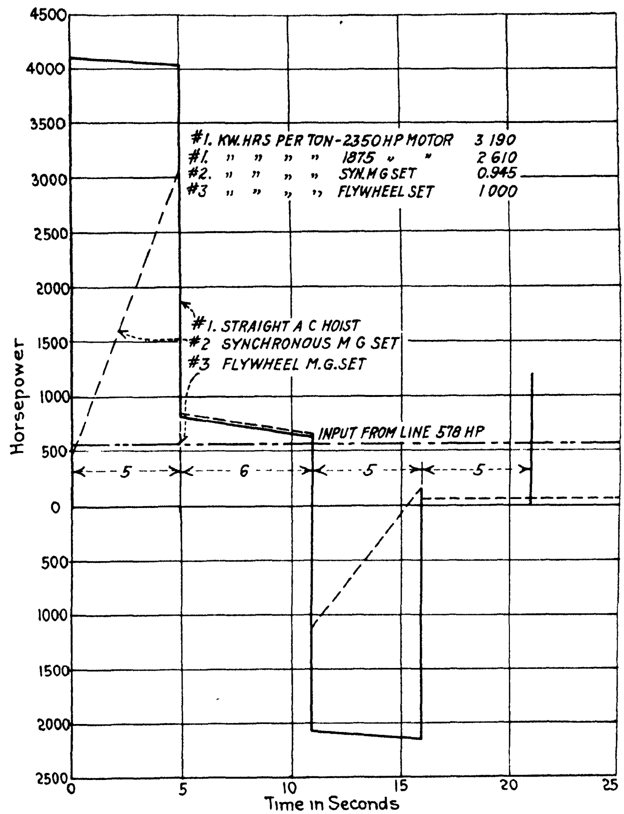


FIG. 11.—CURVE No. 5; INPUT FROM POWER STATION, THREE TYPES OF HOIST.

The capacity of the generator is calculated the same as on calculation sheet No. 5.

The capacity of the synchronous motor is calculated as shown. As there is so much difference in the values during the accelerating and retarding portion of the cycle, the method used is the square of the first value plus the square of the second plus the product of the two and the total divided by 3. The small values at the end of the retarding period

and during the rest period are neglected, as their effect on the heating is negligible. The capacity obtained is 970 hp. A 1000-hp. motor would have sufficient capacity as far as heating is concerned, but would probably not have a sufficiently high pull out. A 1200-hp. motor with a pull out of 3 or a 1500-hp. with a pull out of 2.4, would be recommended. The additional capacity over that required can in many cases be used for power-factor correction. Where power-factor correction is not desired, a simple relay could be installed; this would reduce the excitation during light load periods and increase it during heavy loads, thus tending to produce a more nearly constant power factor and saving some power.

Fig. 11 shows graphically the input to the hoist from the power system with the three types of hoists just described. The heavy solid line shows the input when using the larger of the a.c. hoist motors. The maximum momentary peak at the start is 4100 hp. and the peak during retardation is a little over 2100 hp.; the kilowatt-hours per ton is 3.19.

The input when using the flywheel system is shown by the dot-and-dash line and is practically constant at a value of 578 hp.; the kilowatt-hours per ton is 1.

The dotted line shows the input to the line when using the synchronous motor-generator set. The maximum peak load during acceleration is 3050 hp., but this peak is not imposed upon the system suddenly. The load starts at less than 500 hp. and does not reach the maximum peak for 5 sec.; this permits the regulating devices on the power system to operate and produce good regulaton. All power below the line, when using the synchronous motor-generator set, is returned to the power system. The kilowatt-hours per ton using the synchronous motor-generator set is 0.945, or less than one-third of that required when using the larger of the two a.c. motors.

As before stated, a fair knowledge of hoists and hoisting conditions is necessary to use these forms intelligently, as well as a knowledge of the various pieces of electrical apparatus necessary to make up a complete hoisting equipment. It is necessary in many cases to make compromises and many additional short-cut methods can be evolved by one having to make such calculations daily.

After the machines are determined the details of the control must be settled; this in many cases means conferences with the hoist builders to determine the details so that the electrical equipment will coördinate with the mechanical equipment supplied by the hoist builder.

The control should be so arranged that any wrong or hasty movement of the control levers by the operator will not endanger any part of the hoist. With an a.c. hoist, this is accomplished by automatic acceleration and the well-known safety devices supplied by the various hoist builders. These devices open the power circuit and apply the brakes if the speed is above a predetermined value at any point in the cycle. They also take

care of overwind in either direction. With the d.c. field-control system, the speed of the hoist follows the control lever irrespective of the load and drifting cannot occur; this makes it possible to apply safety devices that would not be effective with an a.c. hoist, also to apply automatic control, which can be made to function without the use of an operator.

With the field-control system the writer prefers to have no switch or circuit breaker in the main armature circuit between the hoist armature and the generator armature, but to obtain overload protection by opening the fields of the hoist motor and generator. A circuit breaker in the main circuit is large and cumbersome and the delay in reclosing it may cause serious trouble, as it is important for the operator to regain electric control of the hoist as soon as possible after the breaker has opened. A circuit breaker located in the field circuit can be more readily handled than one in the main circuit. With the new control system, it is almost impossible to have the circuit breaker opened unless there is some real defect, such as a short circuit in the main generator or hoist motor. This means that the operator has electric control of the hoist at all times, which is much safer than to depend on mechanical brakes in cases of emergency.

DISCUSSION

F. L. Stone,* Schenectady, N. Y.—I think that the author, in speaking of cylindro-conical drums and their use, has created an impression that they are more or less of a fad and not to be considered seriously. I do not believe the author intended to create this impression. There are several types of installations where the cylindro-conical drum has a most decided advantage over the plain cylindrical or straight conical.

Graham Bright.—I did not mean to give the impression that the cylindro-conical drum was a freak drum and had little application, because there are applications where the cylindro-conical drum produces considerable economy; but there are other cases where it should not have been installed and its use has in many instances been advocated under conditions for which it is not well suited.

George S. Rice,† Washington, D. C.—Would those be deep shafts?

Graham Bright.—The cylindro-conical drum is best suited for rather shallow shafts where the hoisting is from one level.

L. F. Mitten,‡ Wilkes-Barre, Pa.—I notice that the calculations were made on the basis of self-dumping cages being used. Is it necessary to take into consideration the unbalanced weight due to the upper cage

* Engineer, Power and Mining Dept., General Electric Co.

† Chief Mining Engineer, U. S. Bureau of Mines.

‡ Commercial Engineer, Vulcan Iron Works.

being held in the dumping guides? In steam-hoist calculations, this is a condition that must always be given consideration. I understand that the motor will have sufficient pull-out torque to take care of this unbalanced condition. The point I want to bring out is whether or not this momentary load would cause sufficient heating to increase the size of the motor.

GRAHAM BRIGHT.— There are cases where this unbalanced weight has made quite a difference in the heating, but in a great many cases the cages can drift into the dump with little or no application of power. The capacity of the motor in most cases is based on a 40° rise. In case the pulling into the dump requires more power than was anticipated, this extra heating is generally taken care of by the fact that the motor has a margin in capacity and can stand a rise higher than 40 degrees.

L. F. MITTEN.—It is not the question of pulling the cage into the dump, but rather that of starting the loaded cage from the bottom with the upper cage wholly or partly suspended by the dumping guides. I made several tests some years ago and found that approximately one-half of the weight of the cage was supported in the guides.

GRAHAM BRIGHT.—The time of getting off the unbalanced position is rather short, and a hoist calculated with sufficient margin (as we generally try to do) will take care of that condition without adding capacity for that particular condition.

L. F. MITTEN.—In other words, the load would not be on long enough to cause additional heat.

GRAHAM BRIGHT —It would cause heat, but not sufficient additional heat to warrant additional capacity.

F. L. STONE.—When the bottom cage has landed and there is some slack rope, either intentional or due to rope stretch, the motor must develop sufficient torque to lift the entire load of ore, car, and cage, completely unbalanced; if the motor will not do this, the equipment may be badly handicapped. Specifications for electric motors for hoists should specify that the motor be of sufficient capacity to make at least one trip completely unbalanced. This, I believe, will take care of all the points raised by Mr. Mitten.

GRAHAM BRIGHT.—That is a point that possibly has not been sufficiently emphasized in the hoist calculations. One thing that is generally taken into account is the making of a complete trip, or possibly several trips, unbalanced. In order to do that, the motor must have sufficient torque and capacity to do what Mr. Mitten and Mr. Stone mentioned, that is, to take care of slack rope systems and starting with the upper cage partly suspended by the dumping horns.

George S. Rice.—In the case of a hoist 3000 ft. or more the weight of the rope would be considerable so I would think that in those cases it would be very important that the spiral drums be used.

Graham Bright.—Out in Butte, where some hoists go down to 4000 ft., they use cylindrical drums, but they also use two or three layers of cable. In that way they are able to use narrow drums with low inertia. If we would use a spiral or a conical drum, all the rope would be in one layer, which means a large diameter drum with a wide face, thus greatly increasing the weight and inertia; so that while something is gained by the coning effect, a great deal of weight is added and also the cost of the hoists greatly increased. These large diameter drums are heavy, and run at low speeds, measured in revolutions per minute. It is necessary to keep the drums as small in diameter as possible, in order to keep the rotative speed as high as possible. The reason for this is that slow-speed electric motors are expensive, so that the smaller the drum diameter the cheaper will be the hoist motor. In deep mines, coning is sometimes used to compensate for rope weight; in shallow mines coning is used to cut down the power required during acceleration and retardations.

F. L. Stone.—On some of the large South African hoists the drum is coned for a considerable distance and then is cylindrical; the rope is usually wound in two layers on the cylindrical portion.

Graham Bright.—Do they taper the rope for the very deep mines?

F. L. Stone.—Tapered ropes have, I believe, been largely done away with. I have not heard of the use of a tapered rope on a new installation for many years. In nearly every case, the drum shape can be so designed as to make tapered ropes unnecessary.

George S. Rice.—What is the opinion of the American manufacturers of hoists regarding the Koepe system, so much used in the north of France and in Germany?

Graham Bright.—They do not seem to be in great favor in this country. The manufacturers and operators seem to feel that the cylindrical or conical type of drums are much safer.

L. F. Mitten.—Possibly I can answer that question. I think there is only one Koepe wheel in this country; of course they are used extensively on the other side. It is my understanding that in the mines in northern France, the Koepe wheels will be replaced with cylindrical or cylindro-conical drums.

Another point that it might be well to bring out is that conical or cylindro-conical drums are not satisfactory where hoisting is done from

several levels. Calculations in all cases should be carefully made if other than a cylindrical drum is used, or bad unbalanced conditions are liable to result when the levels are changed.

GRAHAM BRIGHT.—I should say, in general, the cylindro-conical drum is only suited for one level.

F. L. STONE.—This is also true of the Keope disk. You cannot economically hoist from more than one level with this type of drum.

Mine-water Neutralizing Plant at Calumet Mine*

By L. D. Tracy,† Pittsburgh, Pa.

(Lake Superior Meeting, August, 1920)

On Aug. 5 and 6, 1918, and Mar. 26, 1919, the writer made an investigation of the mine-water neutralizing plant at the Calumet mine of the H. C. Frick Coke Co. The object of this plant is to develop a process of treating the water pumped from the mines so that it may be rendered suitable for use at the power plants of the mines and also at the coke ovens; at the same time, eventually producing a byproduct which will have enough commercial value to provide a sufficient revenue to place the plant on a self-sustaining basis.

The Calumet mine is situated in Mt. Pleasant Township, Westmoreland County, Pa., on a branch of the southwest branch of the Pittsburgh Division of the Pennsylvania Railroad. It is about 6 mi. (9 km.) southeast of Greensburg, the county seat of Westmoreland County.

The coal, which is the Pittsburg or Connellsville seam, lies at a depth of about 200 ft. (60 m.) and is brought to the surface by means of a shaft. The output of the mine averages 200,000 tons of coal annually, all of which is made into coke, either at the ovens, at the mine, or at byproduct ovens. There are 260 coke ovens at the mine.

The continued development of the coal fields of Pennsylvania and the increased use of electric power in the operation of the mines has brought the problem of an increased water supply for the plants to the attention of the coal operators. This is especially true in the Connellsville coking region, where large quantities of water are used in quenching the coke at the ovens. Many of the streams receive the drainage from the mines; and as this water is highly acid and contains sulfur in various forms, some method of treatment is necessary to render it suitable for use at the plants of the mines. With this end in view, the H. C. Frick Coke Co., about 4 years ago, installed at its Calumet mine a plant for experimental purposes, in an endeavor to develop a process that would provide a maximum amount of suitable water at a minimum cost. From a purely technical point of view, the result of these experiments has been encouraging. The company is now endeavoring to place the plant on a commercial basis.

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† Coal Mining Engineer, U. S. Bureau of Mines.

Details of Operation

Calumet mine is drained by three boreholes 8 or 10 in. (20 to 25 cm.) in diameter, and 215 ft. (65 m.) deep, situated possibly 500 ft. from the main shaft. At the foot of the boreholes are four wooden-lined pumps (one 25 in. by 14 in. by 36 in. Lafayette pump; one 25 in. by 12 in. by 30 in., and two 20 in. by 12 in. by 36 in. Yough pumps) which deliver to the neutralizing plant an average of 1,000,000 gal. of mine water every 24 hr. An analysis of a sample of mine water as it comes from the boreholes and before it has received any chemical treatment is as follows:

Analysis of Mine Water

	Grains Per U S. Gallon	Pounds Per 1000 Gal.
Non-incrusting solids:		
Sodium carbonate, Na_2CO_3	None	None
Sodium sulfate, Na_2SO_4	8.5	1.2
Sodium chloride, NaCl	0.9	0.1
Sodium nitrate, $NaNO_3$	None	None
Incrusting solids:		
Silica, SiO_2	3.8	0.5
Ferric oxide plus alumina, $Fe_2O_3 + Al_2O_3$	26.6	3.8
Ferrous sulfate, $FeSO_4$	5.5	0.8
Ferric sulfate, $Fe_2(SO_4)_3$	59 3	8.2
Calcium sulfate, $CaSO_4$	46.6	6.7
Magnesium sulfate, $MgSO_4$	8.3	1.2
Free sulfuric acid, free H_2SO_4	21.3	3 0
Total sulfur trioxide as sulfuric acid, SO_3 as H_2SO_4	165 7	22.3
Suspended matter	14.8	2.1

Description of Plant

Two of the boreholes are located on the opposite side of the railroad track from the main plant. Concrete towers about 18 ft. (5.5 m.) high have been erected over each borehole. Each tower is divided into two compartments, one of which acts as a standpipe into which the water is pumped from the mine below. When the water in this compartment reaches the top of the division wall, it overflows into the other compartment, from the bottom of which a drain leads to a tower between the two boreholes. This tower is similar to those erected over the boreholes, the water filling one compartment and overflowing into the second. From the effluent chamber, a covered concrete drain leads under the tracks and terminates in the mixing chamber of the plant. The entire arrangement is somewhat similar to an inverted siphon. The towers and drain are shown in Fig. 1. The general arrangement of the plant is shown in Fig. 2. The plant is in continuous operation 24 hr. per day; the average operating force, in addition to the superintendent, consists of eleven men.

Pulverizing, Storage, and Conveying Apparatus

The only material used in the process that is not furnished by the raw mine water is the limestone, which is shipped directly to the plant from the quarries in hopper cars and delivered at the outside of the storage shed in the form of screenings that will pass through a ½-in. mesh. The amount of limestone used per day depends largely on the amount of free acid in the water. It is estimated that for every ton of ferric oxide, ¾ ton of limestone is necessary.

These limestone screenings are elevated by a bucket conveyor to a storage bin having a capacity of from 80 to 100 tons. From this bin the screenings gravitate to a Lehigh-Fuller pulverizer, which crushes

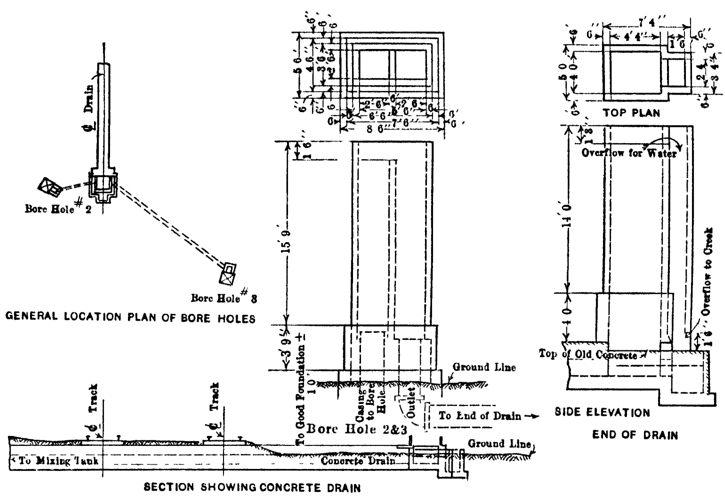


FIG. 1.—PLAN OF CONCRETE TOWERS AND DRAIN.

the limestone so that it will pass through a 200-mesh screen. From the pulverizer, another conveyor carries the limestone to a bin from which it drops on to a screw conveyor, which feeds it to a vertically inclined conveyor. This conveyor elevates the limestone to the mixing tank and drops it into the raw mine water as it comes from the boreholes. The amount of powdered limestone to be used is varied, depending on the amount of sulfuric acid in the water, by increasing or decreasing the speed of the electric motor driving the screw conveyor.

The water from the boreholes is delivered to the mixing tank by means of the inverted siphon arrangement described. As the water enters the mixing tank, the powdered limestone drops into it from the conveyor and is thoroughly mixed as the water passes over and under a series of baffles.

Chemical Reaction

In general, the reactions of the limestones on the mine water are as follows: The powdered limestone, $CaCO_3$, neutralizes the free sulfuric

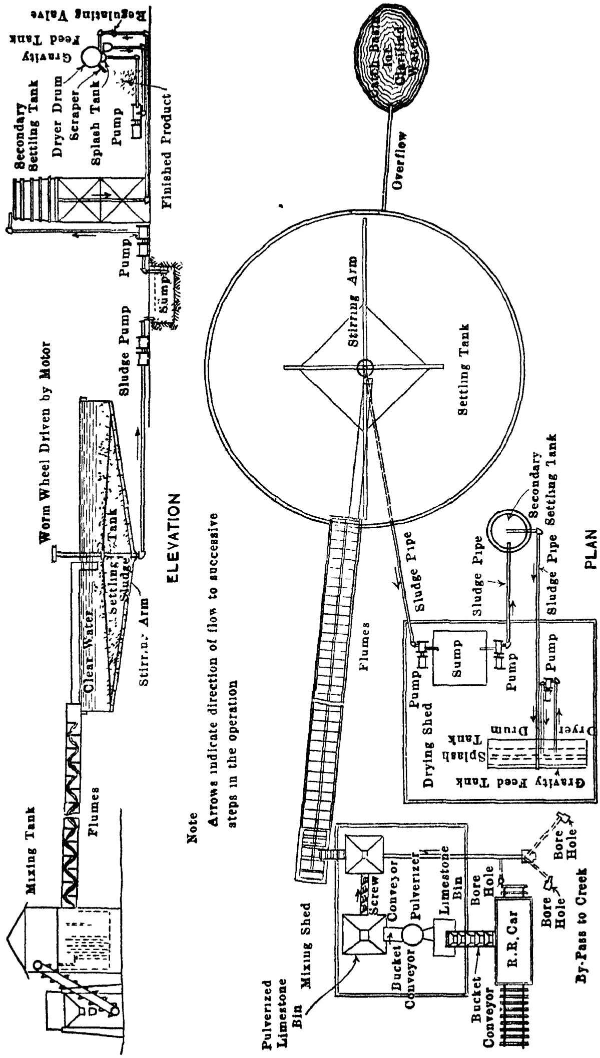


FIG. 2.—OUTLINE OF MINE-WATER NEUTRALIZING PLANT.

acid, H_2SO_4, present in the water, forming calcium sulfate, $CaSO_4$, water, H_2O, and carbon dioxide, CO_2. The basic ferric sulfate is more or less thrown out of solution, since it is held in solution by the acid.

If the free acid is neutralized only, an almost true basic ferric sulfate is the precipitate.

After the free sulfuric acid is neutralized, if an excess of calcium carbonate, $CaCO_3$, is used, the ferric sulfate, $Fe_2(SO_4)_3$, is further reacted upon and decomposed into ferric hydrate, $Fe(OH)_3$, and calcium sulfate $CaSO_4$. However, the calcium sulfate produced is held in solution and does not get into the precipitate in harmful quantities. The resulting precipitate is known as hydrated oxide of iron and is the byproduct of the plant.

Aeration

Leading from the mixing tank to a Dorr thickener is a wooden flume, which is one of the essential parts of the entire process and the design of which is covered by patents. This flume is about 200 ft. (60.9 m.) in length and rests on bents about 8 or 10 ft. high and spaced about 10 ft. (3 m.) apart. It is practically composed of two wooden troughs, side by side, each trough 3 ft. wide and 2 ft. deep. Baffles 2½ ft. apart, alternately projecting from the bottom and from the top, impart to the current an undulating motion, which completes the mixing of the limestone and mine water commenced in the mixing tank and at the same time thoroughly aerates the entire mixture. An arrangement is provided by which any of the pulverized limestone settling in the bottom of the flume may be flushed into a separate tank, and the water drained. Following is an analysis of the treated mine water:

Analysis of Treated Mine Water

	Grains per U. S. Gal.	Pounds per 1000 Gal.
Non-incrusting solids:		
Sodium carbonate, Na_2CO_3	None	None
Sodium sulfate, Na_2SO_4	11.6	1.7
Sodium chloride, NaCl	0.9	0.1
Sodium nitrate, $NaNO_3$	None	None
Incrusting solids:		
Silica, SiO_2	15.5	2.2
Ferric oxide plus alumina, $Fe_2O_3 + Al_2O_3$	56.6	8.1
Ferrous sulfate, $FeSO_4$	8.5	1.2
Ferric sulfate, $Fe_2(SO_4)_3$	None	None
Calcium sulfate, $CaSO_4$	131.2	18.8
Magnesium sulfate, $MgSO_4$	17.4	2 5
Free sulfuric acid, free H_2SO_4	0 5	0.1
Total sulfur trioxide as sulfuric acid, SO_3 as H_2SO_4	172.6	24.7
Suspended matter	100 3	14.3

Suspended Matter

	Per Cent.
Silica, SiO_2	7.6
Alumina, Al_2O_3	4 6
Ferric oxide, Fe_2O_3	41.3
Calcium oxide, CaO	7.0
Magnesium oxide, MgO	Trace
Sulfur trioxide, SO_3	19.1
Water, H_2O	12.4

Dorr Thickener

The flume is built on a 1.5 per cent. grade from the mixing tank and terminates at the center of a Dorr thickener, 7 ft. deep at the outer edge and 12 ft. in the center. When the flume reaches the center of the thickener, it turns down vertically so that the point of discharge is sufficiently below the surface to prevent the agitation of the clarified water.

With a flow through the thickener of 1,000,000 gal. (3,785,332 l.) every 24 hr., the capacity of the thickener will allow a settling period of about 4 hr. During this period, the ferric oxide held in suspension settles to the bottom, while the clarified water passes over the upper edges into collecting troughs, which carry it to a storage basin. From this basin, the water is used if required.

A vertical shaft through the center of the thickener is driven by a worm gear connected to an electric motor. Fastened to the lower end of the shaft are four arms placed at right angles to each other; two are 35 ft. (10.6 m.) long and the other two 16½ ft. These arms are parallel to the bottom of the thickener. On the bottom of each arm, and running diagonally across it, are riveted small steel angles which practically touch the bottom. These scrapers, as they are called, serve to concentrate the settled material at the center of the bottom of the thickener, as the arms revolve at a speed of one revolution every 5 minutes.

A short distance from the Dorr thickener is a building known as the "drying shed." A small diaphragm pump in this shed, connected to the underflow of the Dorr thickener, is used to draw the ferric oxide, or "sludge," from the thickener and discharge it into a sump. From this sump, a centrifugal pump lifts the sludge to an elevated tank outside of the drying shed for a further period of settling. At this stage of the process, the sludge contains about 75 per cent. of water and has the consistency of thick paint.

Drying Process

In the drying shed is a large drum dryer, mânufactured by the F. J. Stokes Mach. Co. of Philadelphia, Pa., which is heated by steam at a pressure of 30 lb. (13.6 kg.). Just underneath the drum, and parallel to its axis, are two troughs, one of which is connected by a line of pipe to the elevated secondary settling tanks. At intervals of about 20 min., a valve in this pipe line is opened and the trough filled. The sludge from this trough gravitates to a small centrifugal pump a few feet in front of the dryer. This pump sends the sludge into the second trough with force enough to splash it against the hot outside surface of the drum. The water is evaporated by the heat as the drum slowly revolves, leaving the residue in the form of a very fine powder, which is scraped from the drum by a long knife-edged steel bar. This powder is yellow in color

and is conveyed by a mechanical loader either to storage piles on the floor or into railroad cars for shipment.

ANALYSIS OF PRODUCT

	Moisture Free Per Cent.
Silica, SiO_2	13.0
Titanium oxide, TiO_2	0 3
Aluminum oxide, Al_2O_3	10 3
Phosphorus pentoxide, P_2O_5	1.0
Ferric oxide, Fe_2O_3	37 1
Calcium oxide, CaO	13.2
Magnesium oxide, MgO	0.6
Potassium oxide, K_2O	1.0
Sodium oxide, Na_2O	0 7
Sulfur trioxide, SO_3	11.6
Combined water, H_2O, above 105° C	4 8
Carbon dioxide, CO_2	6 4
Total	100 0

Changes in Plant

The plant, even at present, is of an experimental nature. Since the original installation, the company has made many changes in the mechanical operation, tending towards greater economy and efficiency, although the process remains the same. The most important change was the elevation of the mixing tank and flume, thereby obtaining a gravity flow to the Dorr thickener, at the same time placing the additional pumping head required upon the pumps in the mines.

In the first design, these pumps forced the water to the surface and it flowed by gravity through the mixing tank and the flume to a sump at the base of the Dorr thickener. From this sump, it was raised by a small centrifugal pump into the thickener. If for any reason the conveyor feeding the pulverized limestone into the raw mine water ceased to operate, more or less of the untreated water found its way to the small pump, with the result that the acid in the water attacked the metal and the pump was soon rendered useless. To overcome this condition, the concrete towers at the bore holes were constructed, the mixing tank and flume were elevated, and the work done by the small centrifugal pump was placed on the mine pumps. As these pumps are wood-lined, the injurious effect of the mine water is reduced to a minimum.

Results Obtained

The principal byproduct of the neutralizing plant is the ferric oxide extracted from the mine water. As has been stated, many of the water courses in this region contain a large amount of sulfur water, which, if used for quenching coke at the ovens, would undoubtedly have a tend-

ency to discolor the coke and increase the sulfur content. It is well known that for blast-furnace purposes the amount of sulfur contained in the coke must be as low as possible. With additional treatment, this water may be rendered suitable for use in steam boilers.

In extremely dry seasons, such for instance as the summer of 1918, the treated water from the Calumet plant has been of material assistance. By using it for quenching coke, the company was able to conserve the fresh water stored in the reservoirs for domestic and power purposes. Before the Calumet plant was constructed, the mine water had been similarly treated in a crude way, when needed for quenching coke.

The possible advantage to be derived from the installation and maintenance of a plant similar to the Calumet plant over the common type of water-treating plants lies in the fact that the former produces a byproduct having a potential commercial value, while the sludge from the latter type has, as far as the writer has been able to ascertain, little if any value. If this byproduct can be made at a sufficiently low cost to meet competition from outside sources of supply of ferric oxide and the plant can be made to produce a revenue sufficient to pay the operating cost, the company will obtain an additional supply of water with little extra cost.

The plant at present is turning out 6 tons per day of ferric oxide. The amount of the byproduct depends entirely on the quantity of water pumped from the mine and the percentages of iron in various forms contained therein. During the war, when the importation of the natural ferric oxide was impossible, a ready market was found for the ferric oxide manufactured. Large shipments of this byproduct were made to the companies manufacturing artificial gas, for use in removing the hydrogen sulfide present in the gas. The problem now presenting itself is to so perfect the operation of the neutralizing plant that, if possible, competition with the European and domestic ferric oxide and iron borings can be successfully met.

Ferric oxide is also one of the chief ingredients in a number of the paints commonly used, and it is hoped that further experimentation will demonstrate ferric oxide to be of considerable value to the agricultural interests of the country and thereby furnish an additional market for the byproduct. The Calumet water-neutralizing plant, so far as the writer knows, is the only plant of its kind in this country which treats mine water in this manner.

Analysis of Various Mine Waters

In connection with experimental work[1] conducted by the Bureau of Mines on the action of acid mine water on the insulation of electric con-

[1] H. H. Clark and L. C. Ilsley: Action of Acid Mine Water on the Insulation of Electric Conductors. U. S. Bureau of Mines *Tech. Paper* 58 (1913) 26 pp.

ductors, the chemical laboratory collected and analyzed a number of mine waters. The acidity and composition of mine waters from the same mine varies considerably at different times. However, the analyses of the samples collected are tabulated here in order to give a general idea of the composition of such waters. The waters tabulated vary considerably in acidity.

MINERAL ANALYSES OF SOME MINE WATERS COLLECTED AT VARIOUS COAL MINES, PARTS PER MILLION

Laboratory Number	Silica	Ferrous Iron	Ferric Iron	Aluminum	Calcium	Magnesium	Sodium	Chlorine	Total Sulfate Radical	Free Sulfuric Acid
10848	36	967	145	60	436	196	85	18	4755	442
10966	40	220	884°	°	162	77	75	25	6370	3662
11785	26	89	299	63	204	130	27	2	2374	93
11786	22	17	12		267	112	100	6	1406	106
11787	16	44	157	25	291	168	65	6	2234	135
11788	32	6	56		234	110	133	Trace	1562	203
11789	20	3	51		79	34	97	Trace	790	248
11790	29	Trace	11		232	118	66	7	1296	88
11791	48	11	915	148	300	197	54	2	4370	Trace
19825	64	6	320	111	295	79	21		2764	277

° Ferric iron and aluminum were not determined separately. 884 represents ferric iron as ferric oxide plus aluminum oxide, $Fe_2O_3 + Al_2O_3$ To convert parts per million to grains per U. S. gallon, multiply by 0 0583. To convert parts per million to pounds per 1000 gal , multiply by 0 00833.

The author is indebted to A. C. Fieldner, supervising chemist, and W A. Selvig, assistant chemist, of the Pittsburgh Experiment Station, for supplying the table of analyses of various mine waters

CONCLUSIONS

The problem of utilizing mine water has been under consideration by mine operators for some time, especially in the anthracite regions, where the quantity of water handled per ton of coal mined is great.

The treatment of mine water has been studied almost entirely from the viewpoint of prevention of damage to pipes, pumps, and boilers, and with a fair degree of success. The principal means of treating impure water used in boilers may be roughly classified as chemical, thermal and mechanical.

The first of these methods seeks, by so-called boiler compounds, either to free the water of mineral salts by precipitation or to remove the free acid by neutralization. The thermal treatment is, fundamentally, a boiling and condensing process and is commonly used when large amounts of carbonates of lime and magnesia are held in solution in water containing an excess of carbonic acid. By heating the water to the boiling point, the free acid is expelled and the salts precipitated. After

sufficient time has elapsed to allow the precipitated salts to settle, the clear water may be pumped into the boiler. The third process is especially applicable to water holding in suspension fine particles of clay, sand, or organic matter. The water is first run into large settling tanks, where the suspended matter is allowed to settle, and then put through a filter, which will remove light organic matter.

Often a combination of these methods has been found advantageous. From the standpoint of the provision of a supply of pure water for boilers, these methods have proved more or less satisfactory. The objections to them are: The cost of the installation and operation of suitable purification plants places an additional burden on the production cost at the mine; none of them has in view the reclamation of any of the valuable elements held in either solution or suspension in the water.

The ideal plant would be one that sufficiently purified the water coming from the mines so that it could be used for domestic or steam purposes, and at the same time produced a byproduct of sufficient commercial value that the revenue derived from its sale would at least partly pay for the cost of the operation of the plant and the interest on the money invested therein. The Calumet plant is a long step toward such an ideal plant. A degree of purification sufficiently high to allow the water to be used at the coke ovens and, with additional treatment, in the boilers is attained. At the same time, about 6 tons per day of a byproduct is produced.

During the long dry spell of the summer of 1918, a number of mines in the bituminous coal regions were obliged to curtail their production for lack of water suitable for use in their power plants. At the same time sufficient water was flowing in near-by streams, which if purified would have kept the mines in continuous operation.

That there are valuable byproducts in ordinary mine water is shown in the operation of the Calumet plant. The ferric oxide is valuable for the purification of artificial gas. Sulfuric acid might be recovered although it can be produced more economically in other ways. However, the value of acid recovered might partly defray the cost of treatment and make it possible to purify water that could not otherwise be treated.

This water also contains some phosphorus and potassium, which may be of value in the fertilization of soils. The water from mines other than coal mines can also be utilized.

The foregoing cases are cited simply to show what might be accomplished when the treatment of mine water is studied from the viewpoint of reclaiming valuable products as well as from the viewpoint of obtaining a pure water supply. It would seem that, in many cases, a plant similar to the one at Calumet might prove to be an economical investment for a company whose surface water supply is in danger of being curtailed in the dry season, for the reason that by possibly producing enough revenue from the byproducts, the cost of operation will

at least be partly met and therefore, will not add to the production cost of the coal. The striking advantage of the Calumet plant is its simplicity of construction and operation.

In the opinion of the writer it would be well worth investigation by any company which has before it the problem of the disposition of mine water and of a poor water supply.

DISCUSSION

CHARLES HAYDOCK, Philadelphia, Pa. (written discussion).—The objects of the plant are stated to be the development of a process of treating the water from mines so that it may be rendered suitable for use at the power plants of the mines, and at the coke ovens, and at the same time produce a byproduct having enough commercial value to place the plant on a self-sustaining basis.

From the standpoint of the first object, the plant is not a success. The analyses of the treated water show the incrusting solids to be many times the accepted maximum permissible scale-forming content of boiler waters. Treating such a water by the customary methods would produce an effluent having such a tendency to foam as to be prohibitive under modern conditions of pressure and superheat. The writer is familiar with the plant and has been advised that the effluent has been used for boiler purposes only when it would otherwise have been necessary to shut down the mine. The harmful effects of using such waters for a short time cannot well be compared with the serious results produced by its regular use. The effluent, however, has been successfully used for quenching coke at such times as the supply of pure water has been inadequate.

As to the requirement that the plant be self sustaining, although no definite figures are given, the results appear encouraging. There is little available literature about byproduct recovery from water-purification plants though it seems probable that future development will be along such lines.

The commonly accepted opinion in the coal fields is that central-station electric service has been most frequently adopted to avoid the difficulties due to using mine waters for boiler purposes, and not as stated by Mr. Tracy. Further, elimination of the power plant simplifies the operation of the mine and, in some cases, substantial savings have been effected by purchasing current rather than generating it in the average mine-power plant, which usually is not of the more economical type.

J. R. CAMPBELL,* Pittsburgh, Pa. (written discussion).—At the top of page 613, Mr. Tracy says that if the free acid is neutralized only, almost a true basic ferric sulfate is precipitated. Strictly speaking, this state-

* Chief Chemist, H. C. Frick Coke Co.

ment is not correct; a modified basic ferric sulfate is the result of either natural or artificial precipitation. W. H. Emmons[2] explains most logically the various reactions that take place in the sulfide enrichment of acid mine water.

A careful examination of the natural precipitate, commonly called sulfur mud, from acid mine water shows that usually it has approximately the following composition:

	Per Cent.		Per Cent.
Moisture	70 50	Alkalies	Not done
Silica	0 70	Iron oxide	63.43
Alumina	3 97	Loss on ignition	31.82
Lime	Nil	SO_3	14.44
Magnesia	Nil		

In this analysis, the iron salts are present approximately as follows: $Fe_2(SO_4)_3$, 24.07 per cent.; $Fe_2(OH)_6$, 71.95 per cent. This may be called a modified basic ferric sulfate, since the hydrolysis, or oxidation, of the ferric sulfate has not been completed according to the last reaction of the theoretical considerations. A true basic ferric sulfate of the formula $2Fe_2\ (SO_4)_3 + 2Fe\ (OH)_3$, has the following composition: $Fe_2(SO_4)_3$, 78.9 per cent.; $Fe_2(OH)_6$, 21.1 per cent. This shows that there is considerable variation from the theoretical considerations. I have called the precipitate from mine water a modified basic ferric sulfate, or a modified ferric hydrate in some instances where the decomposition of the ferric sulfate has been very nearly complete.

The natural precipitate from acid mine water, in an air-dried condition, is valuable for the removal of H_2S from artificial gas, as shown by the following Kunberger tests:

	Dry, Per Cent.	Wet, Per Cent.
Moisture		30.70
Iron oxide	63 71	
First fouling		25.52
Second fouling		23.91
Third fouling		20.06
Fourth fouling		18.63
Fifth fouling		16.02

We take exception to the analysis of the artificial product produced by precipitation with powdered limestone, shown at the top of page 615. This analysis indicates a very high percentage of powdered calcite in the product, also the presence of ferric sulfate instead of ferric hydrate, which is almost inconceivable in view of the fact that gas-purifying material is being produced at the plant. There must have been an error either in sampling or in the analysis. Furthermore, this analysis does not agree with the one shown at the bottom of page 613, which is more nearly correct.

[2] Enrichment of Sulphide Ores. U. S. Geol. Survey *Bull.* 529 (1913) 48.

A number of analyses of the artificial product made at the Calumet neutralization plant, and sold for gas-purifying purposes, have been made:

Analysis of Product	Per Cent.	Probable Combinations	Per Cent.		Per Cent.
SiO_2	12.47	SiO_2	12.47	Sand and clay	23.92
Al_2O_3	11.45	Al_2O_3	11.45		
CaO	7.45	$CaCO_3$	2.27	Calcite	2.87
MgO	0.60	MgO	0.60		
Fe_2O_3	39.43	$CaSO_4$	15.00	Gypsum	15.00
P_2O_5	1.10	$Fe_2(SO_4)_3$	6.20	Modified basic ferric sulfate	55.62
Ignition loss	26.30	$Fe_2(OH)_6$	49.42		
CO_2	1.00				
SO_3	12.54				

The average Kunberger test on the artificial product is about as follows:

	Dry, Per Cent.	Wet, Per Cent.		Wet, Per Cent.
Moisture		18.95	Fifth fouling	19.60
Iron oxide	38.30		Sixth fouling	16.13
First fouling		24.14	Seventh fouling	13.15
Second fouling		24.00	Eighth fouling	12.08
Third fouling		22.92	Ninth fouling	10.06
Fourth fouling		21.46	Tenth fouling	9.58

A complete mineral analysis of a sample of acid mine drainage and the probable combinations are:

Analysis of Acid Mine Drainage	Grains per U. S. Gallon	Probable Combinations	Grains per U. S. Gallon
Total solids	254.07	NaCl	0.96
Chlorine	0.58	Na_2SO_4	13.78
Silica	4.08	$CaSO_4$	68.76
Iron oxide	35.50	$MgSO_4$	24.35
Alumina	10.32	$Al_2(SO_4)_3$	34.57
Lime	28.33	$Fe_2(SO_4)_3$	52.03
Magnesia	8.16	$Fe_2(OH)_6$	19.66
Soda	6.53	Free H_2SO_4	17.13
Sulfur trioxide (total)	128.78		231.24
Total apparent acidity (H_2SO_4)	85.13		
Free acid (H_2SO_4)	17.13		

The iron salts are present in the acid mine drainage as a basic ferric sulfate or a modified basic ferric sulfate. Logically, as Mr. Tracy points out, these iron salts should be precipitated by the neutralization of the free acid present with powdered calcite or some other alkaline reagent. Any excess of powdered calcite will break down the ferric sulfate, as given by Mr. Tracy.

The artificial precipitate is very active and efficient for the removal of sulfuretted hydrogen from artificial gases. The activity and efficiency seems to be due to its alkalinity and colloidal character, it being freshly precipitated material. The precipitate is being used successfully in a commercial way and shows much higher absorption of H_2S than most oxides, like iron borings oxides, bog ore, and other natural ores. In comparative tests, the precipitated oxide showed 40 per cent. more efficiency than iron borings.

If the plant is practicable and economical on a much larger scale than the experimental one described, this process merits the consideration and coöperation of the gas companies. It would seem unfortunate if this valuable byproduct from mine water could not be recovered from acid mine drainage economically, in view of its potential value in the purifying field when artificial gas supplants natural gas entirely, due to its rapid depletion. It is to be hoped that this byproduct from mine water can be developed as surely as the byproducts from coal itself—it really is a byproduct of coal in the final analysis.

W. A. SELVIG,* Pittsburgh, Pa. (written discussion).—The iron that is precipitated from acid mine water on standing, dilution or partial neutralization of the free sulfuric acid is probably a mixture of basic ferric sulfates and ferric hydroxide. Analyses of natural precipitates from mine water show an insufficient sulfur content to form true basic sulfates with the iron present.

Mr. Campbell states that he takes exception to the analysis of the artificial product produced by precipitation with powdered limestone, shown at the top of page 615. The analysis of the sample as submitted by Mr. Tracy was carefully made by a skilled analyst and the writer is confident that it is correct of the sample as submitted.

Mr. Campbell's statement that the analysis in question indicates the presence of ferric sulfate instead of ferric hydrate is hardly justified. Assuming that the hypothetical combinations of the ingredients of the artificial precipitate as calculated by Mr. Campbell from his analysis are correct, the writer by the same method of calculation obtains from the analysis made by the Bureau of Mines: $Fe_2(SO_4)_3$, 7.3 per cent.; $Fe(OH)_3$, 45.8 per cent.

It is rather misleading, however, to state that such a precipitate contains ferric sulfate $Fe_2(SO_4)_3$ as this salt is very soluble in water. The writer prefers to think that the sulfur that may be present combined with the iron is in the form of basic sulfates, possibly $Fe_2(OH)_4SO_4$ or $Fe_2(OH)_2(SO_4)_2$ and that the natural precipitate consists largely of ferric hydroxide $Fe(OH)_3$ together with a smaller amount of basic ferric sulfates.

* Assistant Chemist, U. S. Bureau of Mines.

L. D. Tracy (author's reply to discussion).—In reply to the discussion of Mr. Haydock, the author would say that at no time has he claimed that this treated water was suitable for boiler use. In fact, he says that "with additional treatment this water might be used for boiler purposes." The author agrees with the statement that "Treating such a water by the customary methods would produce an effluent having such a tendency to foam as to be prohibitive under modern conditions of pressure and superheat." But in these days of chemical research it does not seem an insolvable problem to devise methods, other than the customary ones now in use, by which water, similar in analysis to that referred to, may be rendered fit for boiler purposes. There are, however, other uses for water around a power plant than for making steam, one of the most essential of which is for condensation and cooling purposes. In large plants, this is an important item and the use of untreated water, as found in most of the streams, would soon render the steam pipes leaky and unfit for use.

Mr. Haydock states that the central station electric service has been frequently adopted to avoid the difficulties due to the use of mine waters for boiler purposes. There is a tendency at the present time to design large power plants to be located at or near the coal mines, in order to save the transportation costs of fuel. It is a fact that in many of the coal fields the water supply near the mines is more or less contaminated, so that with large central stations, the question of suitable water for boiler purposes is important. In fact, in times of drouth it is necessary to conserve as much as possible the pure surface water so that it can be used in boilers; and by neutralizing this mine water, and thus rendering it suitable for condensation and domestic purposes, an important step in this conservation will have been taken. Whether or not the plant is a success is a question of individual judgment, however, it has been in operation for almost six years under the H. C. Frick Company.

Run-off and Mine Drainage

BY HOWARD N. EAVENSON, PITTSBURGH, PA.

(New York Meeting, February, 1921)

THE eleven mines of the United States Coal and Coke Co. in the Pocahontas coal field are situated in McDowell County, W. Va., which is a mountainous region. The valleys rarely exceed 200 ft. (60 m.) in width, the hillsides slope from 25° to 33°, and the tops, from 600 to 900 ft. above the valleys, carry practically no level land. At least 80 per cent. of the area is covered with timber.

All the workings are drift mines; and while the average grades of the seams worked are ample for drainage, local dips and swamps necessitate the use of a large number of local pumps or of numerous drainage ditches to keep the mines free from water during heavy rains. The water handled in any mine is not great, compared with the coal output, and usually causes little trouble except during extremely wet periods. In new workings, the water encountered, except in rare cases, is trifling; it is only after rib drawing has started and the surface has been broken that water troubles begin.

The strata overlying the coal are sandstone, shales, and slates; as the soil is thin and sandy, it does not tend to puddle and stop the cracks, which remain open for a long time. As mining progresses and the robbed area increases, the broken area becomes larger, increasing the openings into the mined territory. The cover over the robbed portions of the mines varies from 30 to 750 ft. (9 to 228 m.). When the ground is saturated, considerable water finds its way into the mines the day after a heavy rainfall and for three or four days it tends to fill the low places, wash the tracks, etc. and unless from three to four times as many pumps are installed as are normally needed, considerable output is lost. In most cases the water will flow from the mines, and the obvious solution is to provide drainage ditches of ample size to carry the maximum flow expected, so that the inflow will have no opportunity to flood the working places.

Laying out the drainage headings on the proper average grades was an easy matter but where blasting was necessary to provide uniform grades, as was frequently the case, it was important that the maximum flow to be expected and the size of ditches to be provided should be known so that only rock work necessary to provide proper drainage

facilities should be done. The writer was unable to find any data concerning the run-off to be expected in the mines from any given rainfall; these notes are the result of an endeavor to obtain such information.

Rectangular weirs were installed at nine points in different mines, some being so located that the drainage from several mines passed over them. The locations of these weirs, in respect to the mine workings and each other, are shown in Fig. 1. In No. 10 mine, a large swamp gave an opportunity for recording the flow as its area was known and gage readings had been made during a period of extremely heavy rainfall

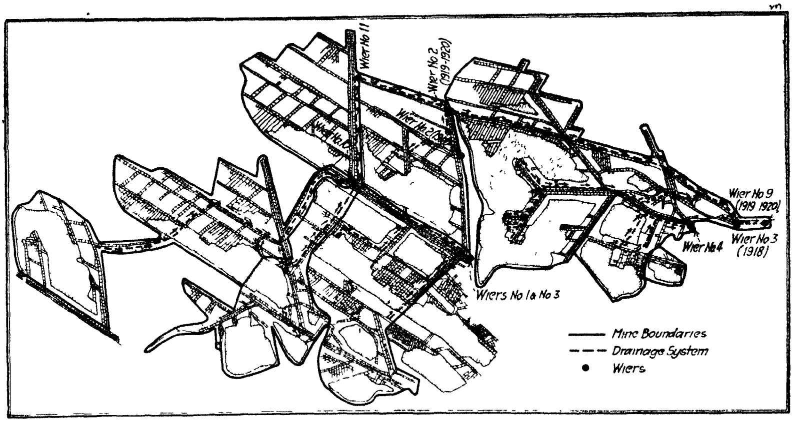


Fig. 1.—Mine-drainage system.

while the pumps were shut down. The drainage of Nos. 2, 6, 7 and part of No. 9 mine flows through a cast-iron pipe at its final discharge, so gage readings from this were also used. These weirs were read at various times, particularly when the flow was the heaviest, and the flow was calculated from them. The Pocahontas Fuel Co. has a long drainage heading through which flows most of the drainage of its Pocahontas mine (the oldest in the field) and its Boissevain and part of the Jenkinjones mines; through the kindness of James Ellwood Jones, general manager and H. B. Wright, chief engineer, weir readings of this flow for two periods are included with the data. Rainfall readings were obtained from the volunteer observing station at Gary; the daily rainfall and the run-off at each weir are shown (Figs. 2 and 3). The average flow, in cubic feet per minute, for each weir was calculated by a planimeter from the run-off curves. The results of these data are assembled in Table 1.

As stated, the amount of water encountered in the first workings is not serious and it was the consensus of opinion that at least 90 per cent.

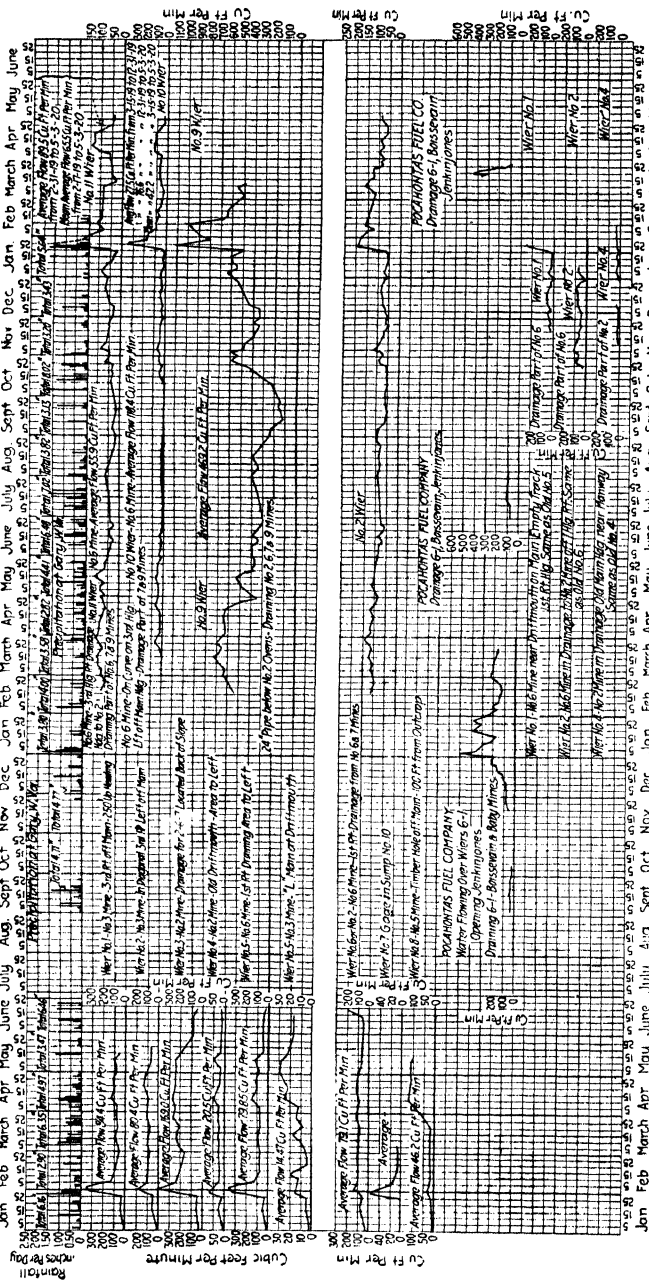


FIG. 2.—MINE WEIR MEASUREMENTS, U. S. COAL & COKE CO.

TABLE 1.—*Drainage and Worked-out Areas and Run-offs—Various Mines*

Weir No.	Mine No.	Drainage Area	Time Covered by Readings	Total Discharge, Cubic Feet	Average Daily Discharge, Cu. Ft. per Min.	Total Rainfall, Inches	Average Daily Rainfall, Inches	Discharge per In. Rainfall, Cu. Ft	Areas Drained: Developed, Acres	Areas Drained: Mined Out: Headings and Rooms, Acres	Areas Drained: Mined Out: Robbed, Acres	Maximum Rate of Flow, Cu. Ft. per Min.	Discharge per Inch Rainfall per Acre, Cu. Ft.	Maximum Flow per Acre Robbed, Cu. Ft. per Min.
			1918											
1	3	Part of No. 3 . .	1/1–5/15	16,373,000	84.4	21.84	0.162	749,680	218.8	16.3	162.9	361	4,602	2.2
2	3	Part of No. 3	1/1–5/20	16,171,100	80.4	21.84	0.156	740,440	184.3	35.8	65.6	194	11,287	3.0
3	2	Part of Nos. 2, 6 & 7	1/1–6/18	41,189,000	169.6	25.59	0.151	1,609,570	1,620.4	247.2	526.4	400	3,058	0.8
4	2	Part of No. 2.. .	1/1–6/19	5,018,400	20.5	25.59	0.151	196,100	155.6	13.4	84.1	64	2,332	0.8
5	6	Part of Nos. 6 & 7.	1/1–6/19	19,547,200	79.8	25.59	0.151	763,860	558.9	84.2	209.2	361	3,651	1.8
6	6	Part of Nos. 6 & 7.	1/1–6/19	19,363,600	79.1	25.59	0.151	756,690	993.4	160.1	284.7	205	2,658	0.7
7	10	All of No. 10.. ..	2/27–28 17–33 hr.	560,000	286.0	2/20–24 2.00	0.400	280,000	500.0	111.0	34.0	286	8,235	8.4
8	5	Part of No. 5 .. .	1/1–5/15	8,966,300	46.2	21.84	0.162	410,550	93.4	5.8	62.2	120	6,600	1.9
			1919–20											
9	2	Part of Nos. 2, 6, 7, & 9.	2/17/19–3/12/20	262,827,000	469.2	57.94	0.149	4,537,920	1,982.1	307.1	778.8	1110	6,515	1.6
10	6	Part of Nos. 7 & 9...	3/15/19–5/3/20	25,218,700	42.2	62.60	0.151	402,850	497.5	86.0	193.2	350	2,085	1.8
11	6	Parts of Nos 6, 7, & 9.	2/17/19–5/3/20	41,689,400	65.5	65.54	0.148	636,090	1,024.8	170.5	377.8	308	1,684	0.8
6'	6	Part of Nos. 6 & 7..	2/17/19–5/3/20	57,665,100	90.6	65.54	0.148	879,850	1,299.0	211.5	479.9	189	1,834	0.4
Jenkinjones		Part of Pocahontas Boissevan.	1/10/16 to 5/29/16	35,528,400	177.5	14.00	0.100	2,537,700		488.0	229.0	355	11,081	1.6
Jenkinjones		Part of Pocahontas Boissevan.	11/20/18 to 3/22/19	31,833,200	181.2	13.14	0.108	2,422,600		569.0	481.0	490	5,037	1.0

of all water handled was due to robbing. As the run-off to be provided for would have to be for the maximum rate of flow, it was concluded to

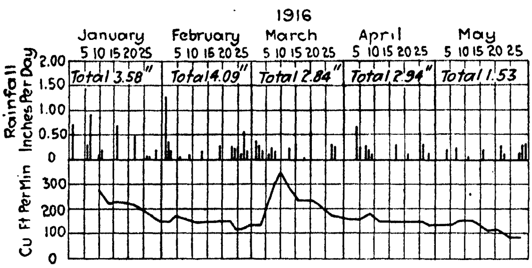


FIG. 3.—PRECIPITATION AT ELKHORN, W. VA.

use only the flow per acre robbed, without paying any attention to the area in open workings.

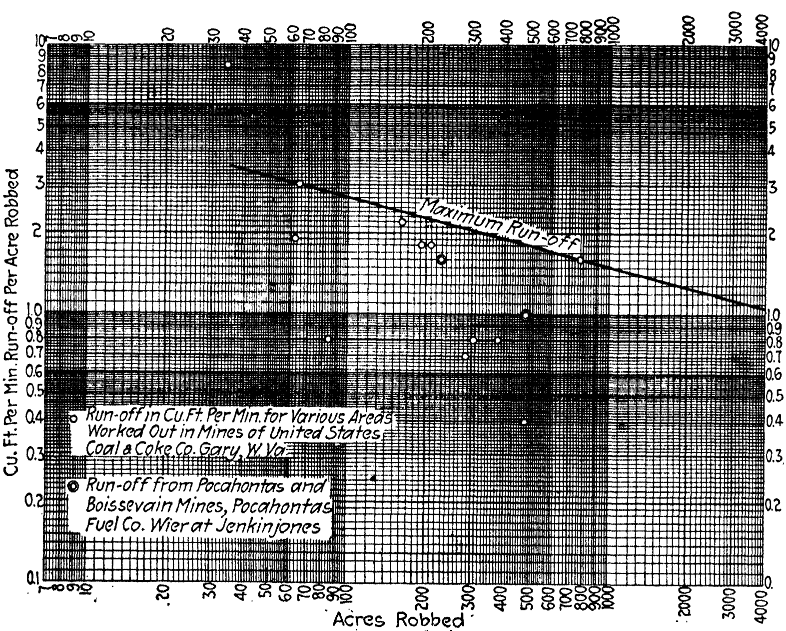


FIG. 4.

Fig. 4 shows the results of all the data plotted on logarithmic cross-section paper with a line showing the maximum run-off from the informa-

tion available. This line is considerably above most of the unit results and is below only one, that of weir 7. The reading here covered a period of only 33 hr. during a period that produced the highest floods in the history of the region, and was for the smallest area covered, so that it is thought that the maximum run-off line is probably more nearly representative of conditions as shown than if weir 7 were considered.

With the probable maximum rate of run-off for various areas determined, Table 2 was calculated. This shows the sizes of the ditches required on grades of 0.5 and 1.0 per cent. for various areas. Bazin's formula for flow in open channels, using the coefficient for exceptionally rough earth channels (as in many cases the ditches will be obstructed by roof falls, debris, etc.), was adopted; for smoother channels, the values given can be greatly reduced.

The areas given will be required when the entire area is mined out. Figures given for run-off are maximum for the areas, according to best data available and should be ample; 11.0 ft. ditch is width of heading. Sizes of ditches can be varied, keeping equivalent area, to suit special conditions. While no attempt was made at scientific accuracy in securing the above data, it is believed that the information is sufficiently accurate for the purpose and that for mines in West Virginia and eastern Kentucky, the sizes of ditches given will be ample.

TABLE 2.—*Ditches Required for Various Mine Areas and Grades*

Area to be Drained, Acres	Max. Run-off per Acre, Cu Ft. per Min.	Total Run-off, Cu. Ft. per Min.	Minimum Grade, 0.5 per cent.					Minimum Grade, 1.0 per cent.				
			Ditch			Velocity, Ft. per Min.	Cu. Ft. per Min. Ditch Will Pass	Ditch			Velocity, Ft per Min.	Cu. Ft. per Min. Ditch Will Pass
			Width, Ft.	Depth, Ft.	Area, Sq. Ft.			Width, Ft.	Depth, Ft.	Area, Sq. Ft.		
50	3.3	165	2.0	1.0	2.0	87	174	1.6	1.0	1.6	105	168
100	2.7	270	3.0	1.0	3.0	100	300	2.25	1.0	2.25	121	272
200	2.3	460	4.0	1.0	4.0	115	460	3.25	1.0	3.25	144	468
300	2.0	600	4.0	1.25	5.0	122	610	4.0	1.0	4.0	153	612
400	1.9	760	4.0	1.5	6.0	136	816	4.0	1.2	4.8	175	840
500	1.8	900	4.0	1.67	6.7	144	964	4.0	1.3	5.2	184	956
600	1.7	1020	4.0	1.8	7.2	145	1044	4.0	1.4	5.6	190	1056
700	1.65	1155	4.0	2.0	8.0	147	1156	4.0	1.5	6.0	198	1188
800	1.6	1280	4.0	2.1	8.4	153	1285	4.0	1.6	6.4	202	1292
900	1.55	1395	4.0	2.2	8.8	159	1399	4.0	1.75	7.0	203	1421
1000	1.5	1500	11.0	1.00	11.0	140	1540	4.0	1.9	7.6	223	1695
1500	1.35	2025	11.0	1.2	13.2	157	2072	11.0	1.0	11.0	198	2180
2000	1.25	2500	11.0	1.4	15.4	175	2695	11.0	1.1	12.1	213	2577
5000	1.15	3450	11.0	1.66	18.2	196	3567	11.0	1.4	15.4	245	3773
4000	1.05	4200	11.0	1.9	20.9	216	4514	11.0	1.6	17.6	270	4752
5000	1.00	5000	11.0	2.1	23.1	224	5170	11.0	1.7	18.7	279	5217
6000	1.00	6000	11.0	2.4	26.4	243	6415	11.0	1.9	20.9	304	6353

DISCUSSION

J. R. Campbell, Pittsburgh, Pa. (written discussion).—I do not recall having seen data on this subject in such detail. In the Connellsville coke region it has been customary to express run-off and mine drainage in terms of tonnage mined; for instance, we figure that from 4 to 7 tons of water are pumped for every ton of coal mined, depending on the local conditions.

The paper says nothing of the character of the water flowing from the mines. However, I take it that this water is generally alkaline in character, or if acid only slightly so. In the Connellsville coke region, the mine drainage is generally acid after development has progressed to any considerable extent. In certain districts, owing to the peculiar stratification above the coal through which the top waters must pass before entering the mine proper, it takes 10 to 15 years for the water to become impregnated with acid and acid salts caused by the oxidation and hydrolization of the ferric sulfide (FeS_2) in the coal.

There are two kinds of acid mine drainage: first, total apparent acidity; second, free acid. Both are expressed in terms of H_2SO_4. The total apparent acidity, which includes the free acid, is due to the presence of certain salts of the metals decomposable by a standard solution of caustic soda (NaOH), using phenolphthalein as an indicator. The determination of total apparent acidity is of prime importance where the water is likely to undergo any rapid process of decomposition, say heating in steam boilers. The iron salts, on decomposition, are very destructive to boilers causing active corrosion and pitting. The sulfates of the metals, principally basic ferric sulfate, decompose and go through a complete cycle in the boilers. The sulfuric acid attacks the metal of the boiler plates and tubes forming iron sulfates, which in turn decompose into oxides and free sulfuric acid.

The amount of free acid in mine drainage, as indicated by a standard indicator, methyl orange, is of lesser importance and cannot be relied on entirely where the water of our rivers is polluted with acid mine drainage, if these waters are to be used as a source of supply to boiler plants. It is customary to determine the free acid in conjunction with the total apparent acidity, the range being usually from 20 to 40 per cent. This range apparently depends on the age of the mine.

The total apparent acidity in the acid mine drainage of the Connellsville coke region is approximately 100 gr. per U. S. gal., consequently the free acid would be one-fourth, or 25 gr. Of course, a number of mines discharge acid drainage ranging up to 400 and 500 gr. of total apparent acidity, but these are exceptions.

Andrew B. Crichton, Johnstown, Pa. (written discussion).—This subject, because of its rapidly growing importance, is causing serious concern in many Pennsylvania communities. While I believe there is

no direct relation between run-off and mine drainage, the larger the mine drainage the smaller is the run-off, and the greater the mine development the larger is the mine drainage. Therefore the people of Pennsylvania should be much more concerned than they are because of the danger of eventually losing important and necessary sources of fresh-water supply. Western Pennsylvania is one of the most important industrial and manufacturing centers of the nation and is probably as densely populated as any similar area; it also is almost entirely underlaid with coal which is rapidly being developed. As old mines become more extensive and new ones are developed, a constantly increasing quantity of mine drainage is pumped into the various nearby streams, until few sources of water supply are not impregnated, or subject thereto, by mine drainage. The drainage from a single mine has destroyed and rendered useless, for all practical purposes, large streams of water, the run-off from many square miles of drainage area.

Many factors affect the amount of mine drainage from a given area, such as the thickness, nature and pitch of the overlying strata, and the surface topography. Some mines have a much larger quantity of water per acre developed than others under as nearly similar conditions as could be found. Of course the shallow mines have the most water, as only a small percentage of our rainfall penetrates deep into the ground. The deep shafts have the least water, save in exceptional cases, where large bodies of water have been tapped in overlying strata by shaft sinking, bare holes, or an occasional fault. In the middle west, due to a thick clay stratum near the surface, water is so scarce in the shaft mines that often water must be taken in for sprinkling.

The statement that 10 per cent. of mine drainage comes from advanced work and 90 per cent. from robbed areas should not be applied generally and would not apply to central Pennsylvania. Generally most water comes from robbed areas, but mines with only a few pillars drawn have developed as much water per acre as other mines similarly situated with large robbed areas. Mines in central Pennsylvania fields produce from 700 to 6000 gal. of water per acre per day, the average over a large acreage being 1400 gal. per acre per day. These figures include both areas worked out or robbed and advance workings where no pillars have been drawn, as it is almost impossible in measuring to separate water from pillars or advance work.

If I understand Table 1 correctly, there is almost the same variance in the mine drainage areas referred to, weir No. 6 averaging 750 gal., weir No. 1, 4185 gal., and weir No. 7, 6160 gal. per acre per day. The West Virginia measurements were taken during what might be called the wet season, while the Pennsylvania figures were in comparatively dry weather.

In an attempt to compare run-off with mine drainage in an area of

about 100 sq. mi., the flow including mine drainage averaged 710,000 gal. per sq. mi. per day. The mined-out coal area was 23½ sq. mi. and the actual measured mine drainage was at the rate of 918,000 gal. per sq. mi. per day, showing conclusively that for each acre of coal mined there is lost considerably more than an acre of fresh-water drainage area. Some mines in western Pennsylvania are pumping from 5 to 20 tons of water for each ton of coal produced. In the Uniontown district, 1½ tons water is pumped for each ton of coal produced; in the Punxsutawney district, 11 tons of water; the average in the bituminous field of Pennsylvania is 4 tons of water. In addition large amounts flow out by gravity.

J. B. Warriner,* Hazelton, Pa.—We have plenty of water in the anthracite region; in the Hazleton region, a few years ago, 16 tons of water were pumped for every ton of coal that came out of the mines. There is little water-level drainage. In some places, drainage tunnels have been driven from the basins to deeper valleys, but, as a rule, the water is handled by pumps or water hoists, though the hoists are gradually being abandoned. The run-off and the water to be handled form one of the most serious problems in anthracite mining; it is more serious than in the bituminous regions, because the possibilities of gravity drainage are less. Pumps used are either reciprocating or centrifugal. The reciprocating pumps have greater efficiency and less power consumption for the quantity of water handled; the centrifugal pumps can be contained in smaller compass with less rock excavation, and with greater capacity for emergency purposes, such as occur at flood times. At times of high water, five or six times the normal quantity of water is handled.

In the Hazleton field, most of the mines are opened under surface drainage channels, such as creeks of quite large size. As the mines are worked out, the water from these channels has found ready access into the mines, so that it has been necessary to ditch the water around these points. In places large canals have been dug outside of the outcrops of the coal measures for the purpose of reducing the inflow into the mines. Directing the flow of the surface water in such a way that it does not get into the mines has been done in this region on a rather large scale, but it has not entirely accomplished its purpose.

R. V. Norris, Wilkes-Barre, Pa.—The works under the heavy wash of the Susquehanna River are usually very dry as we do not dare to break the surface. On the other hand, those near the outcrops are usually very wet because the amount of water coming in is not sufficient to warrant the giving up of the coal to hold the surface.

H. G. Moulton, New York, N. Y.—This paper discusses the water

* Manager Cranberry Creek Coal Co.

which enters the mine from surface run-off as distinct from ground water. Evidently such surface waters can enter excavations either through shafts or openings along the outcrop, or else through cracks in the ground resulting from the subsidence of the surface. Thorough studies of subsidence will result in more knowledge as to the existence of fissures through which water may come, and will also give a clearer idea of the limitations of surface fractures with respect to bodies of water on the surface. Through knowledge of the fundamental principles controlling the location of surface fractures with respect to work below, it would be possible to avoid such subsidences within the danger zones affecting large bodies of water, such as lakes or streams. By knowing in advance where fractures may be expected, it would often be possible to change the course of streams, provide ditches, flumes, etc., and thereby reduce the amount of water entering a mine through the fractures in the overburden. Consequently studies of subsidence may have a direct bearing on the question discussed in this paper.

In certain copper-mining operations, a problem has arisen as to the relation between the area of a stope and the controlling depth beyond which the broken material will arch itself sufficiently to prevent fractures from reaching the surface. One particular stope located directly under a river is now being worked by the square-set method, although other conditions would permit of shrinkage stoping or caving if it were not for the risk of breaking through the surface and filling the mine with water from the river. Questions as to the relation between areas and depths controlling surface subsidence are often involved in the choice of a mining method, when there is danger of letting surface waters into the mine if the surface is fractured.

H. B. Wright,* Pocahontas, Va. (written discussion).—Unfortunately, we have not many analyses of samples of water taken at the outflow weir at Jenkinjones. One sample taken on Nov. 12, 1917, shows:

	Parts per Million	Grains per U. S. Gallon
Total solids at 110° C	1096.0	64.03
Iron (ferrous) sulfate	27.4	1.60
Aluminum sulfate	109.9	6 42
Acidity (to methyl-orange) calculated to sulfuric acid	39 2	2.29
Chlorine	Trace	
Free ammonia	0 88	

A couple of analyses of the water supplying a portion of this outflow at the weir, taken where the greatest destruction to pump was observed, at the time when we were pumping the water instead of draining it by gravity, showed

* Chief Engineer, Pocahontas Fuel Co., Inc.

	Grains per U. S. Gallon
Total solids	119.80
Calcium sulfate	18.30
Magnesium sulfate	6.65
Sodium sulfate	6.25
Sodium chloride	2.04
Iron, alumina, and silica	82.78
Sulfuric acid	3.80
Incrusting solids	107.70
Non-incrusting solids	8.30

Another sample taken from the same place as the above, but at a different time of year, gives:

	Grains per U. S. Gallon
Total solids	42.68
Silica	0.35
Iron and aluminum oxides	1.92
Calcium sulfate	17.26
Magnesium sulfate	9.80
Suspended matter	3 00
Incrusting solids	32.33
Sodium sulfate	9.00
Sodium chloride	1 35
Pounds incrusting solids per 1000 gal	4 61

Hardness equivalent to 20.8 gr. calcium carbonate per gallon.

Alkalinity equivalent to 1.71 gr. of calcium carbonate per gallon.

A sample taken from a portion of the Boissevain mine, when little robbed ground existed, showed:

	Grains per U. S. Gallon
Total solids	11.20
Calcium carbonate	7.25
Calcium sulfate	0.71
Calcium chloride	0.65
Magnesia carbonate	1.89
Free carbonic acid	0.74
Iron, alumina and silica	0 93
Total incrusting solids	11.43
Total non-incrusting solids	0.74

Under certain conditions, as much as 3 gr. of sulfuric acid exists per U. S. gallon, as a maximum, but during periods of flood, this particular element is lacking. The first analysis given would be a fair average for the average flow for the year. Of course, during periods of high water all chemicals shown would appear under analyses in much smaller quantities.

Howard N. Eavenson (author's reply to discussion).—The water from the mines at Gary has always been rather free from acid, but probably this condition will change as the mines become older. The strongest water was from one of the smallest mines and one compara-

tively little developed; why this was so is not clear, as the chemical composition of the coal is the same.

The statement about the proportions of water from advancing work and from robbing would not necessarily apply to other mines. The factors mentioned by Mr. Crichton have a large influence in this, but one of the chief factors is that, in southern West Virginia, almost all of the strata overlying the coal are composed of sandstones and shales, which are not readily influenced by water, and the dirt overlying the rocks is a sandy loam that will not puddle readily and tend to fill up any surface cracks, as will the clay soils of western Pennsylvania.

The correct run-offs for weirs 6, 1 and 7 are as follows: No. 6, 2994 gal. per acre robbed per day; No. 1, 5579 gal. per acre robbed per day; No. 7, 90,605 gal. per acre robbed per day. The West Virginia figures were all taken during wet seasons, the measurement of weir No. 7 being in a short period of extremely heavy rainfall.

The example of the 100-sq. mi. drainage area mentioned by Mr. Crichton is interesting, this working out as a mine drainage run-off of 1434 gal. per acre per day. The other figures given by him for various Pennsylvania districts are, I understand, from the reports of state mine inspectors and should be accepted with caution, as they are undoubtedly estimated, possibly from pump capacities, and cannot be considered accurate.

In considering the point raised by Mr. Moulton, it should be stated that all of the known surface cracks in the vicinity of streams at these mines had been filled and the water ditched away from them or carried over them in flumes. The factors mentioned by him are of great importance in the drainage problem and it is earnestly hoped that the data that is being collected by the Committee on Ground Movement and Subsidence will enable us better to determine where such cracks can be expected in relation to the mine workings.

Steam Regenerators Reduce Coal Consumption

BY W. H. SCHACHT,* E. M., PAINESDALE, MICH.

(Lake Superior Meeting, August, 1920)

IN THE Lake Superior District, the air indoors must be heated continuously during eight months of the year and occasionally during the remaining months. Incident with mining in this district, therefore, there is need for heating many of the buildings connected with these operations. In addition to the mine buildings, that is, shops, engine houses, offices, change houses, etc., such public buildings as schools, library, and theater, when close to the mines, are often supplied from the mine heating mains.

At the Copper Range Company's mines, Champion, Trimountain and Baltic, the heating surface thus supplied from the mine mains is over 50,000 sq. ft. (4645 sq. m.) which requires a steam consumption, at times, of 18,000 lb. (8165 kg.) per hour, or the equivalent of a rate of over 50 tons of coal per day (were live steam used). This figure is based on a consumption of ⅓ lb. per sq. ft. of heating surface during the coldest weather, and provided the system is not wasting steam by blowing it to the atmosphere or into the return line on account of leaky traps, which so often is the case. Under the latter conditions, steam consumptions of three to four times the above rate are not uncommon. This shows that the cost of heating is of enough importance to be given serious consideration, especially at this time of rising prices of fuel.

The power and heating value of exhaust steam are appreciated by most engineers, and its application for such purposes is general. At our stamp mills, the exhaust from the stamps is used to operate low-pressure turbines, thus reducing the former water rate per horsepower developed by more than 50 per cent.

At the above-mentioned mines, electric motors have replaced the many small steam engines, so that this supply of exhaust steam has been practically eliminated. The larger engines are compound, triple, or quadruple condensing. The hoists, however, in all cases are simple and non-condensing and of two sizes, namely, 24 by 60 in. and 32 by 72 in. (61 by 152 cm. and 81 by 183 cm.). These hoists now offer the only available supply of exhaust steam; but as this flow is intermittent and varies from extreme violence to zero, the steam so discharged cannot be

* General Manager, Copper Range Co.

satisfactorily used without some means of storage, such as is offered by a regenerator that transforms the variable flux into a constant one.

Results Obtained by Using Back-pressure Valves

Efforts were made to use this exhaust steam for heating, without using regenerators, by having it discharge directly into the heating system, using a back-pressure valve weighted so as to provide sufficient pressure to permit the circulation of the steam through the system. The greater portion of it, however, for lack of storage capacity, escaped through the back-pressure valve to the atmosphere. The supply so obtained was not sufficient and had to be supplemented with live steam, introduced by an automatic reducing valve at a pressure usually a few pounds under that at which the relief valve was set. This meant that the hoisting engine would exhaust against a back pressure, which (in cold weather and on account of poor design or condition of the system) sometimes was as high as 10 lb. or more, thereby reducing the efficiency of the hoist, because of the reduction of the range through which the steam used by the engine could expand.

With this reduction in efficiency there was a greater steam consumption for the engine, and although some of this additional steam was used in the heating system most of it was discharged to the atmosphere. As a result, the operation was not as profitable as was expected. In fact, in many instances, there was a loss, for live steam fed directly into the the system without the imposed back pressure on the hoist would have resulted in better economy.

The condensate and non-condensable vapors from the several buildings heated were returned to the boiler house and discharged from the system by using steam-driven wet vacuum pumps. This condensate, on account of the oil contained therein, could not be used for feed-boiler purposes and had to be wasted.

Steam Losses Through Use of Radiator Traps

Various types of radiator traps were used in the different systems, some of which were better than others. In a closed system of this kind, by using these traps there is danger of wasting steam into the return line when the traps get out of order. The importance of this fact was not always realized, so that, no doubt, much steam was thus wasted.

Vibration, corrosion, or dirt usually cause these traps to get out of order. If the trap is of the diaphragm or bellows type, the constant vibration is apt to cause fatigue of the metal which, in some makes, results in the loss of the liquid contained therein. When this happens, the valve remains open, causing a continuous flow of steam to the return

line. If the trap is of the expansion-rod type, corrosion or dirt may cause this to stick, with similar results. When these parts give trouble, they are often removed by the workmen who sometimes forget to replace them, so that the waste is permitted to go on.

It is bad practice, also, to discharge main traps (such as are used to discharge from the entire return line) directly into a closed sewer system for, should these traps leak (which they frequently do), the escape of this steam is unnoticed.

Losses Due to Air Leaks

Proper precautions were not always exercised to obtain tight heating systems, that is reasonably free from air leaks, for in some instances two vacuum pumps running simultaneously would not maintain more than 5 or 6 in. (12 or 15 cm.) of vacuum on the return line. This required carrying higher pressures on the main to provide the necessary differential for circulation and resulted in the using of more steam by the pumps than should have been necessary and caused waste due to the imposed back pressure on the hoist.

Much of this leakage was the result of improper provision for expansion of the piping; some was caused by faulty expansion joints and valve-stem stuffingboxes; and some was due to leakage through the rod packing of the hoist. In the metallic piston-rod packing used, steam leakage is prevented by the condensation of the steam, which seals the small passages. With a vacuum on the heating main, there is imposed, also, a vacuum in the hoist cylinder; this difference in pressure causes air to flow into the cylinder through these packings, for there is no condensation to seal the passages. This leakage is considerable and increases with the vacuum carried in the main.

Conditions causing air leakage prevail in all plants, but as long as the steam escaping from the lines is not seen, the system is usually considered to be in good condition by the maintenance man, and the air leaks go unnoticed. The result is that the vacuum pump, which may be large enough to handle moderate leakage, cannot maintain the vacuum, so that the system is operated at a reduced vacuum. This reduction in vacuum is followed by a rise in the working pressure on the main, which rise usually continues until the pressure in the system reaches or exceeds that of the atmosphere. As a result, many persons believe that it is impossible to operate a system at less than atmospheric pressures. Several years of operation at our plants have shown, however, that these engines can be benefited by a vacuum when exhausting into the heating system. This, in reality, is a condenser and, if it is kept tight (as is required of other condensers), there will be no difficulty in operating with main-line pressures of 8 or 10 in. (20 or 25 cm.) provided the steam con-

sumption of the heating system equals or is more than the engine exhausts, while 18 or 20 in. on the return line provide sufficient differential to cause proper circulation under the most severe conditions.

Steam Required to Heat Plant

A survey at that time showed that our steam requirements for heating for more than half of the year were equal to or in excess of that required by the hoists; further, during the coldest weather, at most of the plants, live steam would have to be supplemented, and for the remainder of the year most of the exhaust would be required. Considerable steam is used in the summer months for heating the dry or change houses, water for buildings, and feedwater for the boilers. To use as much of the exhaust steam as possible, it was necessary to provide means for storing this heat and transforming the variable flux into a constant one. For this purpose, steam regenerators of the Rateau design were placed between the hoisting engine and the heating system.

Principles of Operation of Steam Regenerator

The method of treating a flux of intermittent exhaust steam by the use of steam regenerators consists in passing the intermittent flux of steam through a closed vessel, or regenerator, in which the steam is condensed and re-evaporated. Condensation takes place when the steam is in excess of the quantity required to maintain a constant flow of outgoing steam, and re-evaporation takes place when the steam discharged into the regenerator is not sufficient to maintain this constant flow. The condensation and re-evaporation are due to interchanges of temperature between the flowing steam and a large mass of water. In order to keep this water in intimate contact with the steam flowing through the regenerator, the surface of contact must be as great as possible.

In order that the operation of the engine and heating system may be independent of each other, a relief valve located in the steam path limits the range of pressure in the regenerator. This range of pressure, on the other hand, also limits the maximum amount of steam that can be condensed in the regenerator. It is, therefore, essential that the mass of water be brought to the temperature corresponding to the pressure temperature of the flowing steam in order that the maximum storage of heat by the water be obtained before the relief valve blows off.

When the flux of steam received by the regenerator exceeds the amount it is delivering to the heating system, the regenerator absorbs the excess and holds it until some time when less steam is received than is required. When the flux of steam received is just equal to that taken by the heating system, the pressure within the regenerator is constant. No

steam is condensed and the regenerator floats idle. Should the quantity of the incoming steam be less than that required by the heating system, the pressure in the regenerator falls, and the water gives off steam, losing the heat that was absorbed during a time of excess.

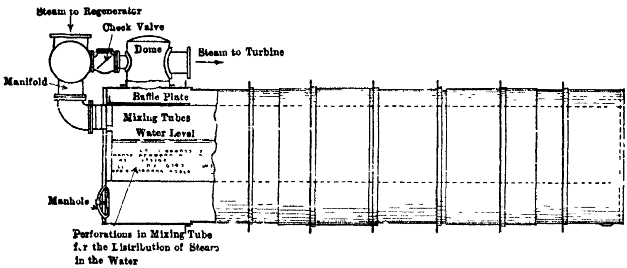


FIG. 1.—LONGITUDINAL SECTION, RATEAU PATENT STEAM REGENERATOR.

The efficiency of the regenerator as a piece of power apparatus is, as usual, the ratio of power output to power input. If the regenerator delivers steam at the same pressure as that at which it receives steam, barring the slight losses due to radiation, its efficiency is 100 per cent On

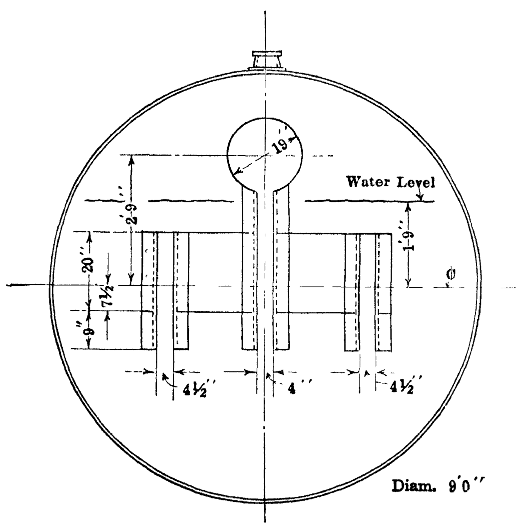


FIG. 2.—CROSS-SECTION OF REGENERATOR.

the other hand, if the regenerator always receives steam at a higher pressure than that at which it can deliver steam, its efficiency is less than 100 per cent., and the loss is represented by the pressure drop between the absorbing condition and the condition of delivery.

The regenerator is equipped with a suitable steam dome for the discharge of the regenerated steam to the heating system, and with a special automatic water-level device, which removes the water of condensation entrained with the exhaust steam. A relief valve controlling the maximum pressure permissible between the exhaust of the engine and the main inlet is located at any suitable point of the piping. The exhaust steam discharged into the regenerator always contains water. After passing through the water in the regenerator, the steam never contains more than a fraction of 1 per cent. of moisture.

This apparatus was found to be equally efficient as an oil separator. The condensate of the regenerated steam is practically free from oil and

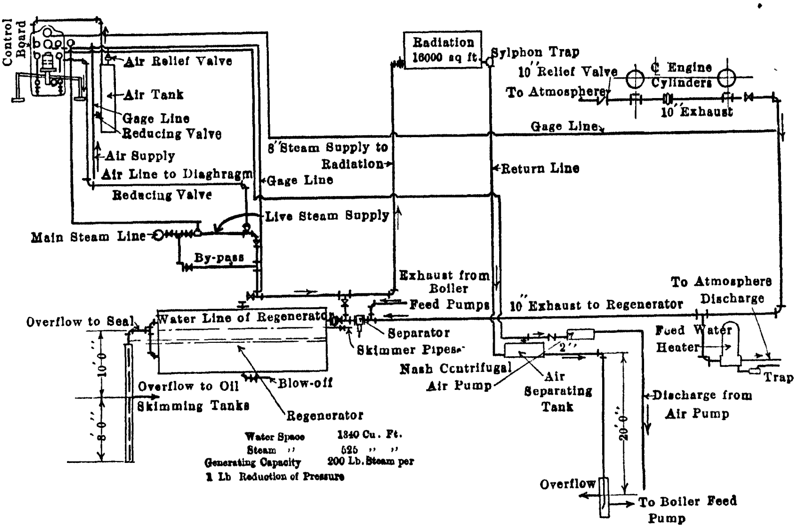


FIG. 3.—DIAGRAMMATIC SKETCH OF PIPING SYSTEM.

suitable, therefore, for boiler feed, which was not the case before regenerators were installed. A longitudinal section and a cross-section of the regenerator used are shown in Figs. 1 and 2.

PIPING SYSTEM, GAGES, AND VALVES

The buildings to be heated are located, in some instances, 2300 ft. from the engine house, and well above the elevation of the boiler house, thus offering ideal conditions for gravity returns and good drainage. As a result, the condensate is returned to boiler house and discharged from the system (operating at 20 in. of vacuum) by means of a water seal. Because of the effectiveness of the regenerator as an oil separator, this condensate is used for boiler feed.

In remodeling the several heating systems, estimates of the steam requirements were made and pipe sizes used that would give a pressure drop through the mains of approximately 1 lb. The return lines installed were also of ample size, so as to cause no undue pressure loss. Advantage was taken of the difference in elevation and the pipes so installed as to eliminate all pockets, thus permitting the entire system to drain back to the boiler house, avoiding, therefore, possibilities of trouble when steam is shut off from the system in freezing weather.

Asbestos sponge felt 1 in. (25 mm.) thick was used where the pipe lines are sheltered, that is, where they are suspended in concrete boxes, and 1½ in. thick was used where the pipe lines are exposed to the weather. The return lines are also covered so as to conserve this heat for boiler-feed

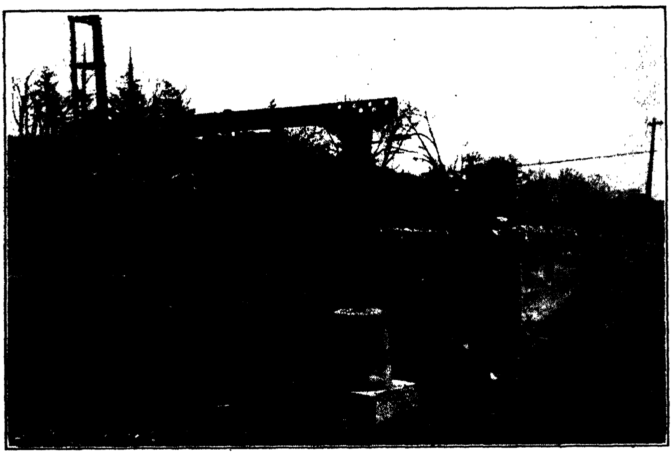


Fig. 4.—Method of supporting pipe lines.

purposes. These coverings are protected from the weather by asphalted asbestos jackets, which when in place are given a coat of asphalt paint.

No slip expansion joints were used, expansion being provided for by offsetting the lines 50 or 60 ft. (15 to 18 m.) at intervals of 500 ft. The piping is usually anchored midway between the offsets. One of the systems at the Champion mine, which has 2300 ft. (701 m.) of mains in which all expansion is provided for in this manner, has not required the use of a wrench on it since it was installed about three years ago.

These pipe lines are suspended in long slings, which are free to swing from an eyebolt having a vertical thread adjustment for pipe alignment. Some sections of these lines span deep ravines, requiring, therefore, bents 30 ft. high in order to avoid pockets and to provide the necessary run for drainage. A diagrammatic sketch of one of these installations is shown in Fig. 3, the methods of supporting the pipes is shown in Figs. 4, 5, and 6.

The exhaust from the hoist cylinders is piped to the regenerator inlet, which is placed at least 6 ft. (1.8 m.) below the cylinders to avoid drawing water from the regenerator back into the cylinders when the engine is idling or drifting, thus causing a vacuum sufficient to raise water several

FIG. 5.—A TANGENT CROSSING A DEEP RAVINE, CARRIED ON BENTS, IN SOME PLACES OVER 30 FT. HIGH.

feet above the water level in the regenerator. In some installations the regenerators are placed in the basement of the engine house; in others they are placed in the boiler room. An atmospheric relief valve in the

FIG. 6.—EXPANSION OF THE TANGENT TAKEN UP WHERE IT EMERGES FROM THE BURIED BOX. LONG HANGERS ARE NEEDED, AS A TRAVEL OF 12 OR 13 IN. IS REQUIRED.

exhaust line near the engine can be set to maintain any pressure desired but it is usually set to relieve at ½ lb. above atmosphere. The exhausts from the boiler feed pumps, hoist auxiliaries, and trap discharges from the hoist receivers are also piped to the regenerator.

Connected to the exhaust line between the engine and the regenerator is the feedwater heater, which gets its supply of steam from the engine or pump exhaust, or draws from the regenerator when the other supply is insufficient. In other words, the feedwater heating is not dependent on an intermittent supply, but has a constant supply to draw upon, and temperatures of 190° to 200° are easily obtained. As the pressure in this line is below the atmospheric, a power or steam operated steam trap is used to discharge the condensate from the heater. Considerable condensation or entrained water is carried with the exhaust steam into the regenerator, which must be discharged from it. As the working pressure carried is usually below the atmospheric and as ample head can be obtained for a water seal to discharge under these conditions, this method

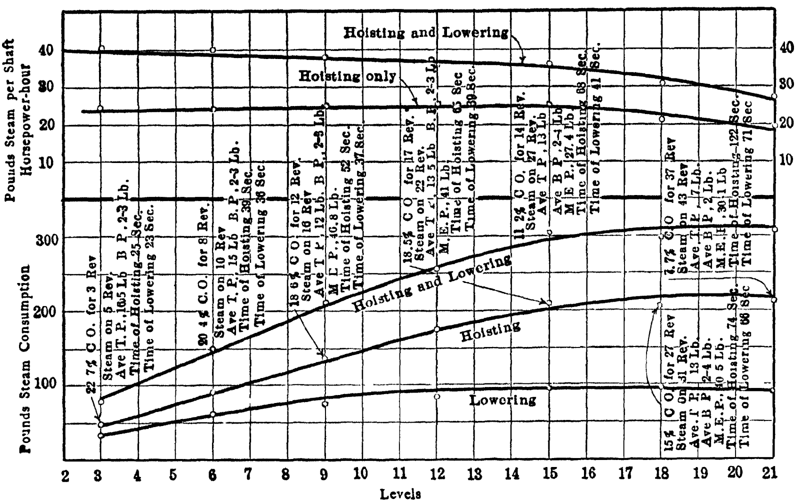


FIG. 7.—STEAM CONSUMPTION FOR HOISTING AND LOWERING, NO. 1 SHAFT. TWIN 32 BY 72-IN. SIMPLE ENGINES ON ATMOSPHERIC PRESSURE. ROCK LOAD 5 TONS.

of relief is used. The overflow of the seal is situated so as to operate either with vacuum or pressure, being able to discharge to the atmosphere with pressure of 4 or 5 lb. (1.8 to 2.3 kg.) plus or minus.

Provision is made to supply live steam to supplement the exhaust automatically. Two reducing valves are used in series, to safeguard the system in case one of these valves should get out of order. By stepping down the pressure in stages in this manner, the cutting action of the steam on the valve and seat is much reduced.

The reducing valve on the high-pressure side is of the ordinary type and reduces down to 15 lb. (6.8 kg.) The other valve is of the diaphragm type operated by auxiliary pressure introduced by means of a special control mechanism. An ordinary reducing valve, depending on the

pressure of the atmosphere, will not operate on pressures when less than atmospheric. A telltale pressure gage, which is tapped into the line between the reducing valves, is used to indicate when either of these valves is out of order. Should the gage read above 15 lb., the valve on the pressure side is not tight and may require reseating; should the gage read less than 15 lb. the valve on the lower side is not tight.

These reducing valves automatically admit steam to the system at any predetermined pressure, either above or below atmospheric. This is done with a special control mechanism, made by the Klipfel Manufacturing Co., of Chicago, which consists of a closed chamber or cylinder, the lower end of which contains a flexible rubber diaphragm to which is attached a stem or yoke that connects with a lever arm fulcrumed to one

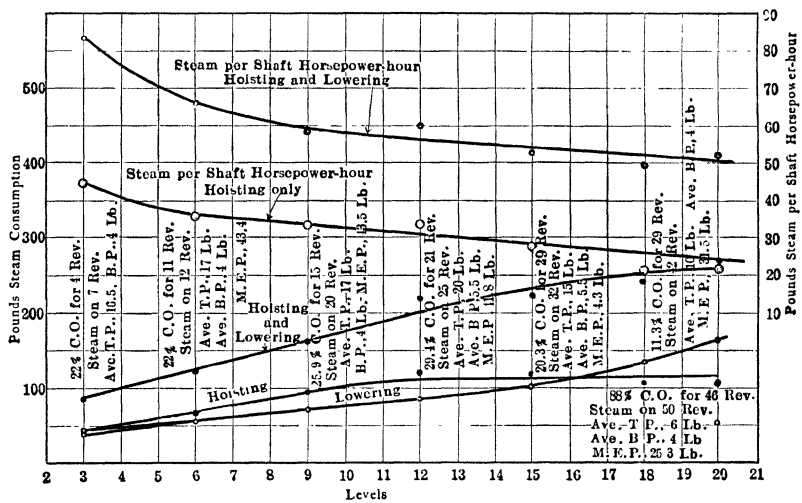


FIG. 8.—STEAM CONSUMPTION FOR HOISTING AND LOWERING, NO. 3 SHAFT. TWIN SIMPLE 24 BY 60-IN. ENGINES. ROCK LOAD $2\frac{1}{2}$ TONS.

side of this connection and capable of being weighted on either side of its fulcrum. This, in turn, makes contact with the stem of a valve connected to a supply of air under pressure, usually 15 lb. The chamber is connected to the steam space of the regenerator and also serves as a connection for a special gage for indicating pressure carried on the regenerator. This pressure causes a deflection of the diaphragm, which, in turn, acts upon the lever arm. The weights can be adjusted on the lever arm so as to balance against any pressure in the regenerator. For instance, if steam is to be admitted to the system at 4 lb. minus pressure, the weights are so adjusted as to permit the lever to be moved sufficiently by this pressure to operate the air valve, causing it to open and admit air to the diaphragm of the reducing valve. This, in turn, opens and admits

the make-up steam to the system, staying open until pressure rises to the 4 lb., when the action of the lever is reversed and the steam shut off.

For the operation of the low-pressure reducing valve, a supply of compressed air at mine pressure is stored in a large receiver that is connected to the mine air main. Leather-seated check valves prevent the return or loss of the air in storage when the pressure in the main is reduced or the compressor shut down. The storage is sufficient to operate these valves for 48 hr. The air taken from these receivers is reduced to a pressure of 15 lb. by means of a reducing valve and used at that pressure by the control board.

The steam from the regenerator is delivered to the buildings to be

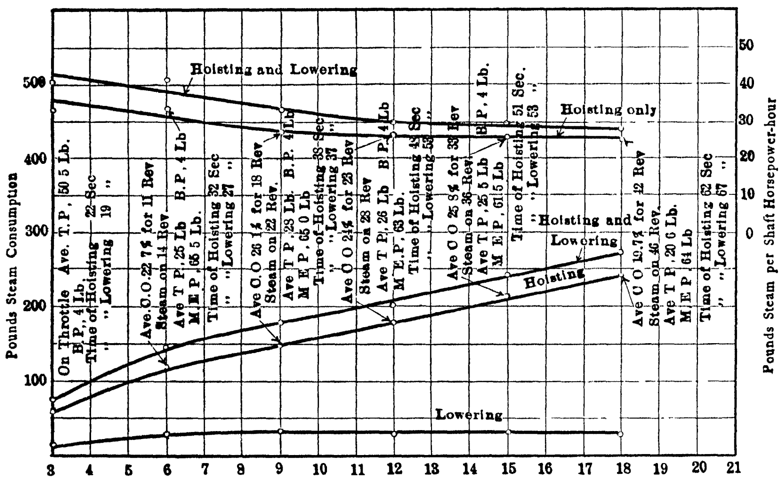


FIG. 9.—STEAM CONSUMPTION FOR HOISTING AND LOWERING, WITH ENGINES ON ATMOSPHERIC PRESSURE, NO. 4 SHAFT. TWIN 24 BY 60-IN. SIMPLE ENGINES. ROCK LOAD 5 TONS.

heated with a drop in pressure not to exceed 2 lb. at times of greatest demand and about 1 lb. during average winter conditions. A two-pipe system is used. In the buildings, the drainage of mains and return lines is in the direction of the flow, thus eliminating all water hammer. Sylphon traps are used on all radiators, direct and indirect, and on all headers and drips.

The air and condensation are collected into and returned by a 3-in. line, the end of which is submerged, forming a water seal that is located in the boiler house. At a point about 20 ft. (6 m.) above the overflow of this water seal is located an air separating tank, from which the air and non-condensable vapors are drawn from the system by a Nash No. 0 vacuum pump, requiring for its operation about 1½ hp. This pump is operated by an electric motor. We have operated some of these pumps for several years, during which time their maintenance costs have been small.

The pressure carried on the regenerator, being also the back pressure on the hoist, varies from 8 to 10 in. vacuum to ½ lb. above the atmosphere; the colder the weather, the lower is the average of the range maintained. The return-line pressure is usually constant at about 18 in. (45 cm.) making a working pressure available, therefore, of about 8 lb. (4 kg.) which is more than sufficient in almost any emergency.

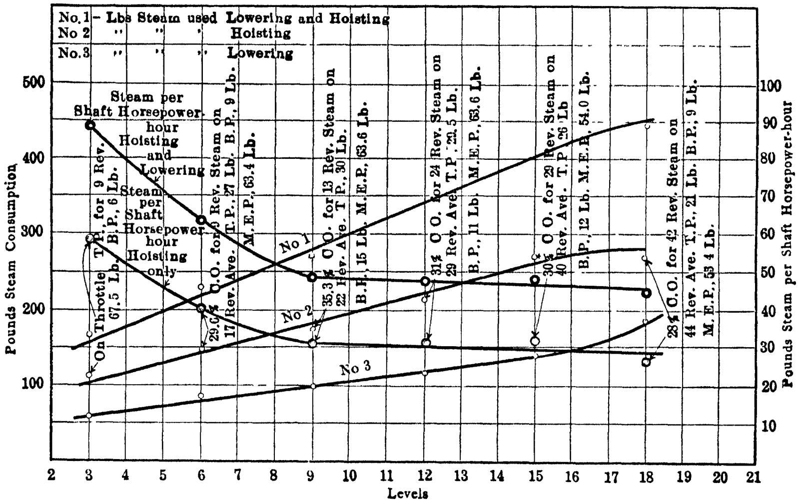


FIG. 10.—STEAM CONSUMPTION FOR HOISTING AND LOWERING, NO. 4 SHAFT. TWIN 24 BY 60-IN. ENGINES. NO. 1 TEST. ENGINE ON HEATING SYSTEM. L. H. ENGINE INDICATED. ROCK LOAD 5 TONS.

RESULTS OBTAINED BY USE OF REGENERATORS

The coal consumption of the several plants before and after installing regenerators, also the tons of rock hoisted during same periods and the radiation involved in each plant are given in the accompanying table.

TABLE 1.—*Results Obtained by Installation of Regenerator*

Shaft	Coal Consumption				Rock Hoisted				Radiation Involved
	Before Regenerator Installation		After Regenerator Installation		Before Regenerator Installation		After Regenerator Installation		
	Years	Tons	Years	Tons	Years	Tons	Years	Tons	Sq. Ft.
Nos. 2–3, Champion..	1916	5,340	1917	2,076	1916	290,000	1917	240,000	18,000+
No. 4, Champion.....	1916	4,296	1917	2,256	1916	335,000	1917	292,000	11,000+
No. 2, Trimountain...	1918	4,235	1919	2,908	1918	303,650	1919	295,510	7,000+
Nos. 2–3–4, Baltic....	1918	11,410	1919	9,766	1918	373,150	1919	349,970	11,000+
		25,281		17,006		1,301,800		1,177,480	47,000+

There was a slight reduction in the tons hoisted in most plants during the year following the regenerator installations, but this did not affect the coal consumption materially, for, in most cases, some live steam was used for heating, there not being enough hoisting to supply the need. At Nos. 2 and 3 Champion, the live steam thus used would have been available for additional hoisting, so that only a few more tons of coal would have been used had the same tonnage of rock been hoisted. The results exceeded our expectations and, in some instances, the installation and remodeling costs were returned by the savings made during the first year's operation. In all cases at least one boiler was shut down, and in some, two were discontinued.

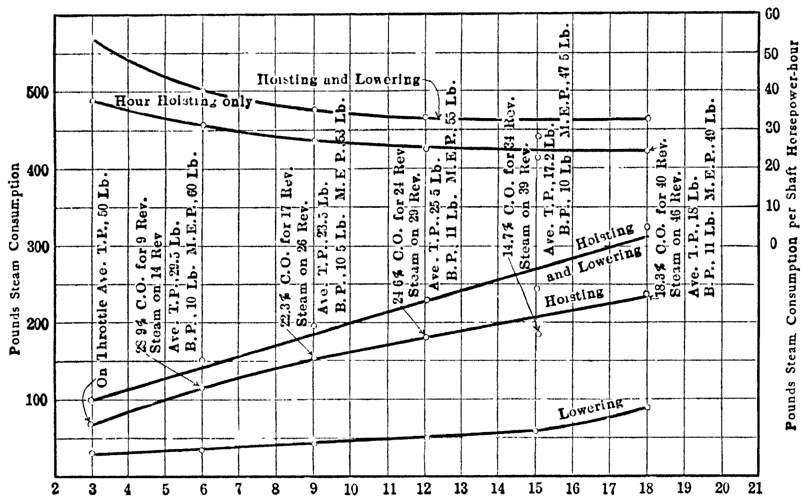


FIG. 11.—STEAM CONSUMPTION FOR HOISTING AND LOWERING, NO. 4 SHAFT TWIN 24 BY 60-IN. SIMPLE ENGINES. ROCK LOAD 5 TONS.

In the forepart of the paper it was stated that the exhaust from some of the hoists was used for heating before regenerators were installed and that back pressures of 10 lb. or more had to be carried at times to supply the necessary steam for heating. It was desirable to know the relative steam consumption of the hoists operating under these conditions as compared with their operation when exhausting to the atmosphere. We were concerned, not so much with the actual, but rather with the relative consumption of steam, so steam-indicator diagrams were used. It was assumed that the condensation in the engine cylinders and pipe lines under both conditions would be the same, so that the diagram should reveal, approximately, the actual difference.

The skip, in each case, was filled with what was considered an average load of rock; this load was used throughout the test for all hoisting. Continuous indicator cards were taken, hoisting from every third level. From these diagrams, the average cut-off pressure and the length of cut-off were determined and the weight of the apparent steam consumption ascertained. These tests were made with various back pressures on the engines, and were run on a 24 by 60-in. (61 by 152 cm.) engine, using a 5-ton skip, at No. 4 shaft of the Champion mine; a 32 by 72-in. engine, using a 5-ton skip, at No. 1; a 24 by 60-in. engine, using a 2½-ton skip, at No. 3; and a 32 by 72-in. engine, using a 2½ ton skip, at No. 2. The revolutions made by the engines were counted for each run, and the time in which runs were made ascertained by stop watch. The mean effective pressure,

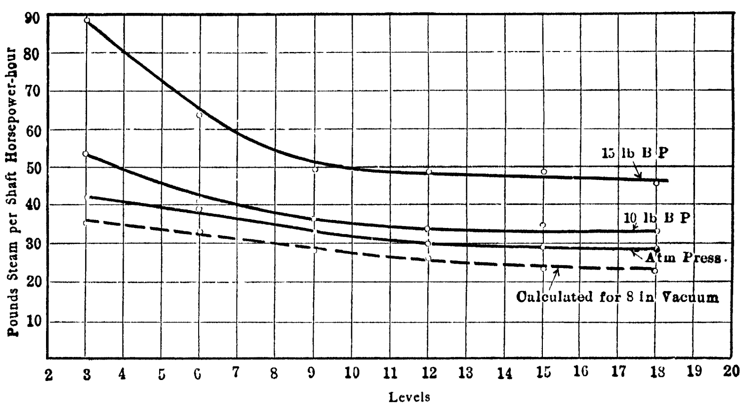


Fig. 12.—Steam consumption as determined per shaft horsepower-hour for hoisting and lowering with 24 by 60-in. engines. Rock load 5 tons.

total steam consumption, and pounds steam per shaft horsepower-hour were determined for both hoisting and lowering from the various depths. No account was taken of the steam of condensation, the amounts represented are as determined by the diagrams.

Curves showing these values are shown in Figs. 7 to 12. Fig. 12 makes a comparison of the steam consumption per shaft horsepower-hour for a 24 by 60-in. engine hoisting a 5-ton loaded skip when operating at various back pressures. The dotted line shows the calculated consumption were the hoist to operate at 8 in. vacuum, which is the back pressure obtainable when regenerators are used; this result has since been verified. This comparison shows why a saving of one-third of the coal is possible.

Method Used for Calculating Regenerator Requirements

To show method of figuring regenerator requirements, the requirements for the large plant at Champion, viz., No. 2 and 3 have been taken as an example.

The radiation to be heated at that time totaled 16,000 sq. ft. Allowing 1/3 lb. of steam per square foot for coldest weather, the maximum requirements per hour would be 5300 lb., and for average winter conditions, with a consumption of 1/4 lb. per sq. ft., 4000 lb. per hr. In addition to this, about 600 lb. are required for heating boiler feedwater, and 10 per cent. must be allowed for condensation in transmission, requiring, therefore, approximately 5000 lb. per hour as the average consumption. The hoisting averaged about fifty-five trips during the 8-hr. working period, and was from an average depth of 1600 ft. The time to make the complete hoisting cycle was 110 sec. The idle period, or time during which no steam flows, was 6.9 min., and under actual conditions at that time did not often exceed 12 minutes.

The exhaust from two hoists are used to supply the regenerator. The average steam consumption of these engines per cycle when hoisting from the sixteenth level (this being the average hoisting depth) was found to be 275 lb., making available an average of 3800 lb. per hr. To this must be added steam from the auxiliary cylinders of the hoists, which amounted to an additional 12 per cent. or about 400 lb.; also, 200 lb. from boiler feed pumps, making a total available of 4400 lb. per hour.

For a 12-min. idling period of both hoists, a steam storage of 1200 lb. was required to provide for the heating requirements during the coldest weather. The amount of water required to store that amount of steam, when using a pressure range on the regenerator from 8 in. vacuum to zero, can be determined as follows:

Latent heat of steam at 4 in. of vacuum, or average of the working range = 975 B.t.u.

975 × 1200 lb. steam equals 1,170,000 B.t.u.'s required.

Temperature of water in presence of steam at atmospheric pressure, 212°; temperature of water at 8 in. of vacuum, 197.7°; difference, 14.3°.

Pounds of water required to store this heat = 1,170,000 ÷ 14.3 = 81,818 lb.

Weight of water at the above temperature equals 60 lb. per cu. ft.; amount of water required, 81,818 ÷ 60 = 1363 cu. ft.

Storage capacity of the regenerator installed, 1190 cu. ft.

DISCUSSION

O. P. Hood,* Washington, D. C.—As the original equipment at these mines was the most efficient available, when it was installed 20

* Chief Mechanical Engineer, U. S. Bureau of Mines.

years ago, to be able to save about 30 per cent. of the fuel by this installation is a notable achievement. In 1909, a regenerator equipment was installed by the Quincy Mining Co., but it was a failure because it could not meet the conditions as to back pressure. In the present case, the back-pressure problem has been solved apparently by turning the whole heating system into the condenser. For each pound of increase in back pressure in such a hoisting engine, about 3 per cent. more steam is required. This was shown in the tests of the regenerator installations at the Quincy and in the Carnegie Steel Co's. plant in Pittsburgh, the results of which agree with those found by Mr. Schacht. There are few regenerators so installed that they can be tested. Mention is made of four systems; could anything be gained by tieing these systems together? Is there a decrease in the temperature of the steam served to the buildings so that increased heating surface is needed?

W. H. Schacht.—The No. 2 and 3 hoists at the Champion mine exhaust into one regenerator, thus using the steam from two hoists. The several systems, however, are not interconnected, as the advantages gained, under present working conditions, would not justify the cost. However, should one of these systems have more steam than it requires and the others have an insufficient supply, such a connection may be justified.

While there is a decrease in the temperature of the steam now used, additional radiation is not necessary. Much of the heating surface, prior to the remodeling of the several systems, was inactive owing to air-bound radiators. Proper drainage of return lines resulted in the removal of this air, which more than offset the drop in temperature.

During the war, the state representative in rating the plants graded them according to the appliances used. A plant having CO_2 meters, etc., was rated higher than one without them, other things being equal. Plants having modern appliances in the boiler rooms, such as we have at the stamp mills, were rated as high as 98 per cent., although the exhaust from these stamps may have been wasted to the atmosphere at a pressure of 30 lb. Yet, because our mine plants used locomotive firebox boilers and did not have CO_2 meters and means for weighing coal, they were rated at 60 per cent. No credit was given for utilizing the exhaust by the use of the regenerator, which resulted in saving, in some instances, over 30 per cent. of the coal.

Lynch Plant of United States Coal and Coke Co.

By Howard N. Eavenson, C. E., Pittsburgh, Pa.

(Wilkes-Barre Meeting, September, 1921)

Early in 1917, the United States Coal & Coke Co. secured options on several tracts in Harlan County, Ky., aggregating about 19,000 acres in area, and after careful prospecting by outcrop openings and diamond drilling, completed the purchase late in July of that year. This property is situated in the eastern end of Harlan County, south of the Poor Fork of Cumberland River, and extending across Big Black Mountain to the Clover Fork of Cumberland River, a distance of about 7 mi. (11.3 kg.). It extends from the Kentucky-Virginia line westward for about 6 mi. Looney Creek, which empties into Poor Fork, crosses the property for about 4 mi. (6.4 kg.), and as several of the seams within the property outcrop along this stream, it afforded an easy, and the logical, place for development.

The property is on the northern side of the geological trough formed by the uplifting of Stone Mountain, on the southern, and Pine Mountain, on the northern side; the northern boundary along Poor Fork is practically at the base of Pine Mountain. This area has been described in various geological reports.[1]

The main seam on this property is the one known variously as the "C," Benham, Keokee, Taggart, and Roda; it is also called, by the Kentucky Geological Survey, the Kellioka seam of the western portion of Harlan County. This seam averages, in this property, about 5 ft. (1.5 m.) in thickness, although local rolls reduce this thickness considerably over small areas, and at the extreme western end of the property the seam splits. It is usually clean, although occasionally a small parting occurs within a few inches of the bottom of the seam; it is one of the best coking and gas coals in the United States. The average analysis of this coal is given in Table 1.

[1] Philip N. Moore: Report on Iron Ores in Vicinity of Cumberland Gap. Geol. Survey of Kentucky [2] 4, Pt. 5.

J. B. Dilworth: Black Mountain Coal District, Kentucky. *Trans.* (1912) **43**, 129.

Kentucky Geol. Survey· Upper Cumberland Coal Field. *Bull.* 13 (1912). Supplementary Report on the Coals of Clover Fork and Poor Fork in Harlan County (1916).

TABLE 1.—ANALYSES OF COALS FROM C SEAM

Numbers of samples	24	Coke	70.1
Volatile matter, per cent	35.09	Ammonium sulfate, lb	25.0
Fixed carbon, per cent	60.47	Benzol, lb	29.2
Ash, per cent	4.44	Tar, gal.	7.5
Sulfur, per cent	0.59	Gas, cu. ft	11,448
Phosphorus, per cent	0.005		

The purchase negotiations were completed late in July and about Aug. 4 the decision was made to start the development of the tract at once and to push it with all possible speed, as the Government wanted an additional supply of high-grade, high-volatile byproduct coal from which the production of benzol and toluol could be increased. Actual possession of the property was given about three weeks later and on Aug. 26 construction work was started and pushed with all possible speed until after the armistice was signed.

There was available a good survey of the property lines, but nothing that showed any topography or the relation of the coal outcrops to the property lines, and the only time available for securing these data was the three weeks between the two dates mentioned. A topographic party was organized, and the work of getting the necessary information for the townsite layout was pushed as rapidly as possible, using a transit survey for the base lines, and filling in the topography by plane tables. The scale used was 1 in. = 100 ft. (0.025 = 30.5 m.) and the contour interval was 10 ft. (3 m.). The site available for the town, all of which will ultimately have to be used, covered an area about 2½ mi. (4 kg.) long, and in places, about 2000 ft. (609.6 m.) wide. The topography for the plant site layout was not complete until the early part of November, 1917, and the layout map was finished a few days thereafter.

When directions to begin the construction of the plant were issued, the output wanted was 2,500,000 net tons per year. The property could not be attacked, except from the Looney Creek site, without considerable railroad construction, and as this site was in the approximate center of the tract, and all of the coal could be reached from it with a maximum haul of 4½ mi. (7.2 kg.), and an average haul of about 2 mi., it was decided to construct at this point a plant that would have a capacity of not less than 8000 tons per day. For this output two mines were projected, one on each side of Looney Creek; the pit mouths were placed a short distance above high-water mark, so that as much of the coal as possible would drain to them. The coal from both mines was to be taken to one tipple, at which the entire output would be loaded. It was realized that such a plant would be the largest single loading plant in the United States, perhaps in the world, and its construction gave opportunities for concentration in the line of shops, wash houses, tipples, amuse-

ment buildings, housing facilities, etc. that were unique, and which enabled the company to build units and facilities of a size that are ordinarily unheard of in the usual coal-mining community.

Housing

The section in which this plant is located is entirely isolated from any large community; in fact the only town of any size is the neighboring plant of Benham, which contains perhaps 2000 people. There is no other community of more than a few people within 20 mi. (32 kg.) by rail, or 9 mi. (14.5 kg.) across a steep mountain trail. Everything required for comfortable living in the ordinary large town would have to be furnished by the company, so it was fortunate that all of the property was owned in fee simple, and that there were no outside lots or owners to create embarrassment in laying out the town.

It was estimated that, to produce the capacity required, the average daily output by employees of all classes would be about 4 tons; so for the desired production of 8000 tons per day, 2000 men would be required for all purposes. At the older plants of the United States Coal & Coke Co., in the Pocahontas field, a careful census, taken at various times, had shown that ordinarily there were two people on the plant for each man on the payroll. It was estimated that at Lynch, partly because of its location and partly because so many of the foreign miners had left the country, this small percentage would not hold, but that the probable population would be about 6000 to 7000.

The number of rooms per man employed in coal-mining communities in isolated situations of this kind has been steadily rising for some years. In the Pocahontas region, at the older plants, the number of rooms has risen within 6 years from 0.99 to 1.85 rooms per man; so it was decided that two rooms per man should be built, or a total of 4000 rooms. As the most popular type of house averages four rooms, this meant a total of 1000 houses.

The amount of space available for various purposes, while larger than is usually the case in this mountainous region, was not adequate for a town of this size, using single houses; even with double houses it was necessary to place the houses closer together than would otherwise be desirable. The final decision was to construct 400 double houses and 200 single ones, and to place the houses at a minimum distance of 30 ft. (48 m.) apart. At many of the older plants, the need of a large clubhouse or hotel has been badly felt many times. Prior to starting the Lynch plant, it had been the custom at many of the large plants to provide a fair-sized building in which the clerks, engineers, store employees, and some of the unmarried mine officials could find suitable living accommodations. After a careful study of the conditions and requirements at

the older plants of the company, it was decided that a building having at least 100 bedrooms, together with the necessary dining rooms, parlors, toilet rooms, etc., would be required for the Lynch plant; the building erected contains 133 rooms, of which 108 are bedrooms, in addition to a large basement and attic; the remainder are the usual service rooms. There has been considerable criticism of the size of this building, but at practically no time since it was completed have there been any unoccupied rooms. In fact, during much of this time it has been necessary to provide sleeping quarters in the attic. The basement was utilized for a shoe shop, clothing store, jewelry shop, soda-water fountain, and cigar store; four bowling alleys and three pool tables also were installed. In addition, five large boarding houses were built for the accommodation of the single miners. The number of houses built is shown in Table 2.

TABLE 2

Plan Number	Type	Number of Rooms	Bath	Number of Houses	Total Number of Rooms	Number of Families
Supt. 46	Single	8	Yes	1	8	1
Officials						
33	Single	7	Yes	6	42	6
40	Single	6	Yes	6	36	6
41	Single	5	Yes	6	30	6
32	Single	4	Yes	7	28	7
37	Single	3	No	18	54	18
38	Single	3	No	42	126	42
39	Single	4	No	95	380	95
35	Single	5	No	15	75	15
42	Single	23	Yes	5	115	5
14	Double	8	No	182	1456	364
29	Double	8	No	58	464	116
36	Double	10	No	30	300	60
34	Double	12	No	10	120	20
31	Double	6	No	120	720	240
				600	3954	1000

The various types of houses are shown in Figs. 1 to 7. They are all plastered, except a small number built the first winter (which are ceiled with wall board) and are of wooden construction, except six officials' houses, which are concrete block, stuccoed. All of the plant buildings and about fifty of the houses are heated by hot water. These have solid stone-wall foundations; the remaining houses are heated by grates, one in each room; are set on stone piers, and have double floors with building

Fig. 1.—Community House and Dwellings.

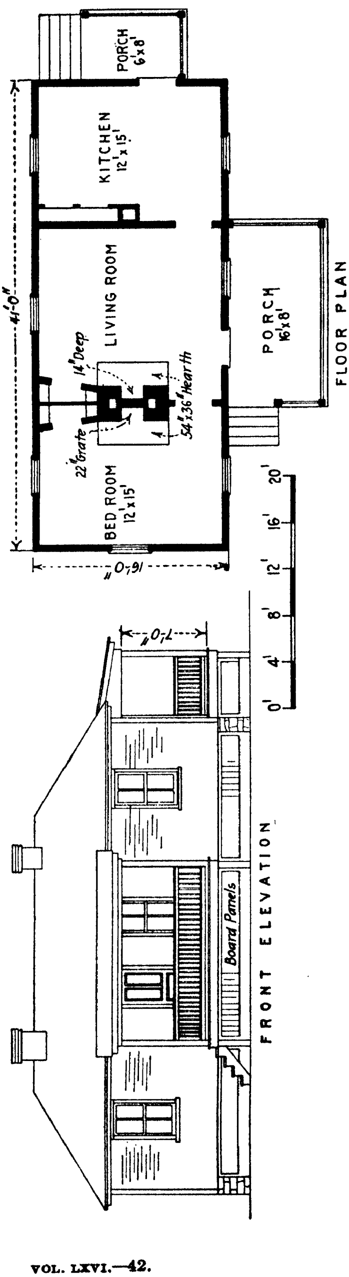


FIG. 2.—SINGLE THREE-ROOM HOUSE.

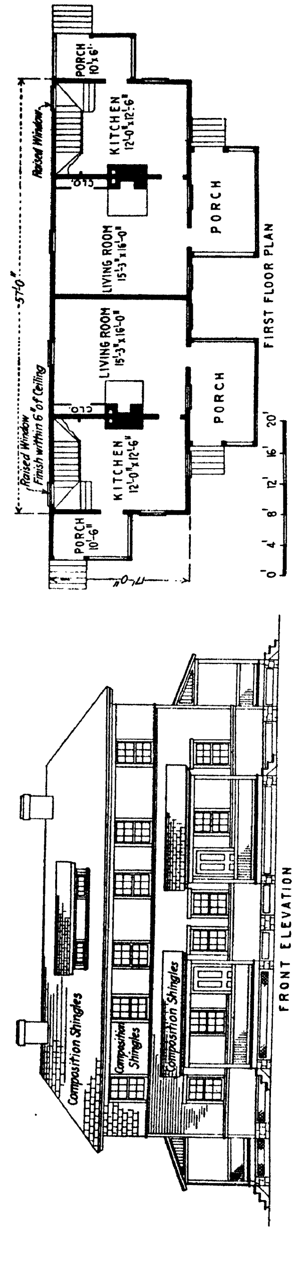


FIG. 3.—FOUR-ROOM DOUBLE HOUSE.

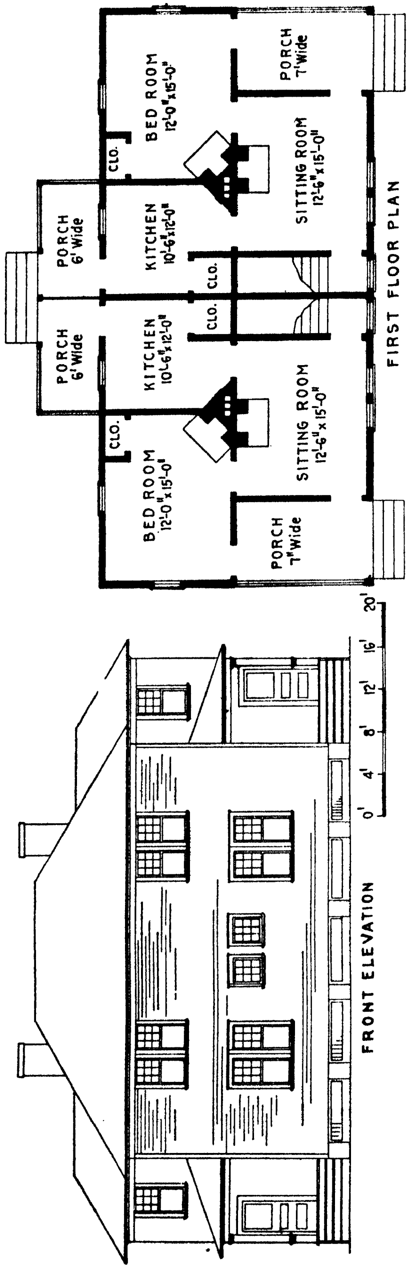


FIG. 4.—DOUBLE FIVE-ROOM HOUSE.

paper between, for the lower floor. Spaces between piers and between porch posts are closed by flooring boards. All roofs, and the gables and upper stories of some of the houses, are covered with asphalt shingles,

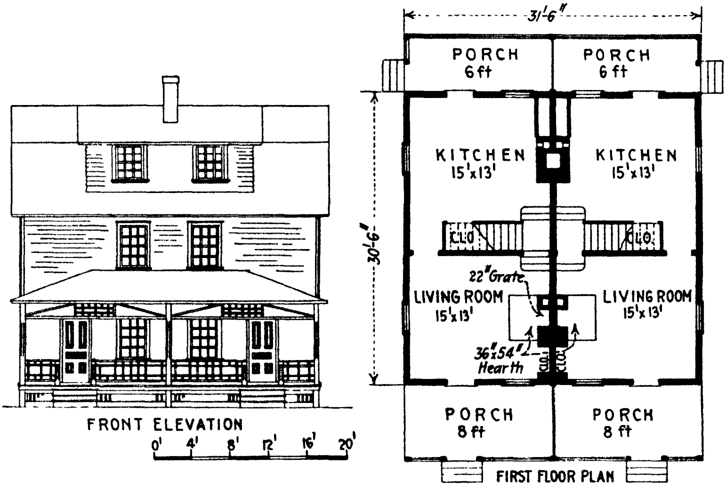


FIG. 5.—DOUBLE SIX-ROOM HOUSE.

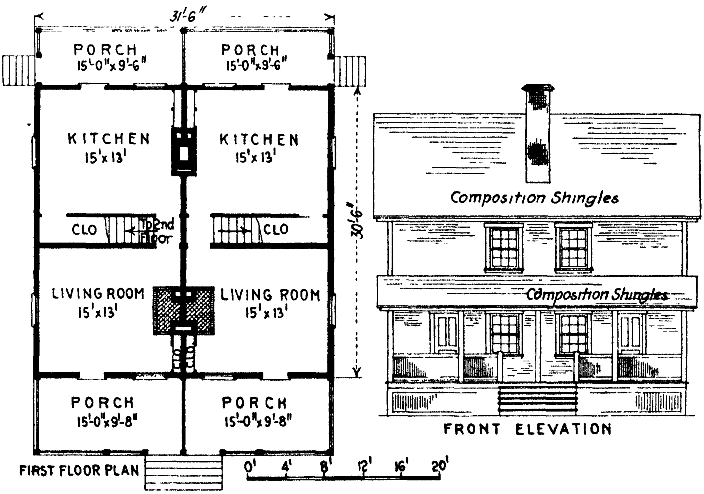


FIG. 6.—EIGHT-ROOM DOUBLE HOUSE.

both green and red being used. Experience has shown that, taking both appearance and wearing qualities into consideration, this type of roof is the most satisfactory cheap covering yet tried.

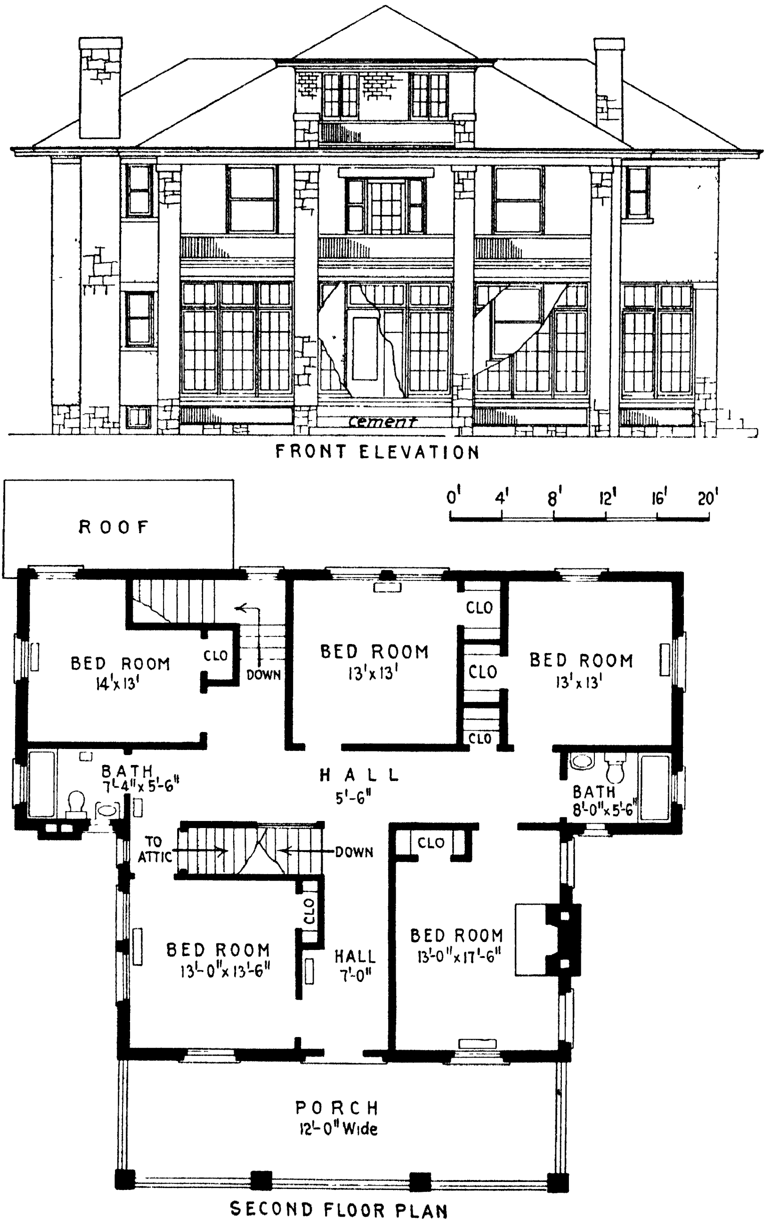


FIG. 7.—SUPERINTENDENT'S RESIDENCE.

In many of the house types, it has been possible to vary the details and shapes of the house and porch roofs of houses having the same interior arrangement. These changes, the number of plans used, and the various colors used, have prevented the monotony in appearance usually seen around coal-mining plants. All houses are given three coats of ready mixed paint in the outside and two on the inside. Yellow, light gray, and white body colors are largely used, though a small number are painted green and red. White, green, and black are used for trim.

Lots

Where the ground is flat, the lots are usually 100 ft. (30.5 m.) deep and vary from 32 to 66 ft. (9.8 to 20 m.) wide, depending on whether the house is single or double. On the hillsides, a broad slightly sloping bench about 43 ft. (13 m.) wide is excavated, a road and sidewalk being placed at the bottom of the slope, just back of the house. The lots used allow gardens as large as the ordinary family cares to cultivate, although they would have been made larger had space allowed. All houses are fenced with a standard steel-wire fence; in this climate this type of fence has proved very durable and satisfactory.

Streets and Alleys

The town layout was made so that no grades on the streets exceeded 10 per cent. at any place; on the main street none over 5 per cent. were allowed. The main street throughout the town is a part of the state highway system and a roadway width of 22 ft. (6.7 m.) with a 4-ft. (1.2 m.) walk on each side was provided. On the side streets, the roadway was made 16 ft. (4.9 m.) wide with the same sidewalks. Alleys were made 10 ft. (3 m.) wide. All sidewalks, curbs and gutters, walks from the sidewalk to the front and rear porches of each house, and the entire roadway of Lynch road were made of concrete; all are one-course work, 1:2:4 mix, and built in accordance with standard specifications of the Portland Cement Assn. It is the intention to macadamize all side streets, as the traffic on them will be comparatively light.

Water Supply

Experience at the older mines had shown that a supply of 50 gal. per person per day would be sufficient for town purposes, so a steel tank with a capacity of 300,000 gal. or about 1 day's supply, was installed on the hillside above the power house, at an elevation sufficient to supply pressure for two standard fire streams at the highest fire hydrant in the town. The supply mains are 8 in. (20 cm.) in diameter, and the distributing lines are 8, 6 and 4 in. in diameter, and so interconnected that a supply can be furnished practically anywhere in the town through two lines. All pipes are steel, asphalt coated, with sleeve joints. All fittings are flanged;

flanges are also put on all lines at intervals of about 200 ft. (61 m.). Every line has a valve where it leaves the main and blow-out valves are provided at dead ends. The house supply lines are all 1 in. (2.5 cm.) asphalt-coated pipes, and in each kitchen a cast-iron sink about 20 by 30 in. (50 by 76 cm.), is installed. Each service pipe is provided with a stop and waste cock underground with a handle extending into the kitchen; between the ground and the floor it is boxed and surrounded by manure. Fire plugs are placed at intervals so that no house is more than 200 ft. from at least one plug, and waxed canvas fire hose, 2½ in. in diameter, with 1 in. nozzles in 250-ft. lengths, on reels, are provided at suitable intervals.

The water supply is procured from deep wells, located near and above the power plant. These are blown by compressed air and the water flows to a concrete sump at the power plant, from which centrifugal pumps, having a capacity of 800 gal. per min., lift it to the tank. The pumps are in duplicate, but only one is used at a time. The well supply has not been sufficient, so lines have been laid up Gap Branch and Looney Creek to points well above the town, from which the creek water will flow to the sump, where it will be chlorinated before being pumped to the tank. In exceptionally dry periods, the creek supply is also quite limited, so provision is being made for a storage supply by driving some rooms in the mine under Gap Branch, which will be filled, through a borehole, in the spring and the water be retained by a dam until needed, when it will flow by gravity to the sump at the power plant.

Lighting

The town is lighted by electricity, the main distribution lines to transformers being 6600 volts, and from the transformers on the three-wire system 230–115 volts. All houses have at least one light in each room and one on the porch. Where the houses are close to the street, the porch lights give enough illumination; in other places, street lights on poles are used.

Sewage System

Except the main buildings and houses having inside toilets, all houses have outside closets equipped with concrete septic tanks. These are constantly kept filled with water and the fœcal matter liquefies and escapes through 4 or 6 in. (10 or 15 cm.) sewer pipes, to which also the kitchen sinks drain, into the main sewer. The entire town is sewered and at its lower end the sewage flows into a concrete sump, from which centrifugal pumps lift it to a treating plant on the hillside, Fig. 8. Here the sewage passes through septic tanks, a settling basin, and a chlorination tank, from which it flows to the creek. Provision has been made for a sand filter, after the chlorination treatment, but this has not been built

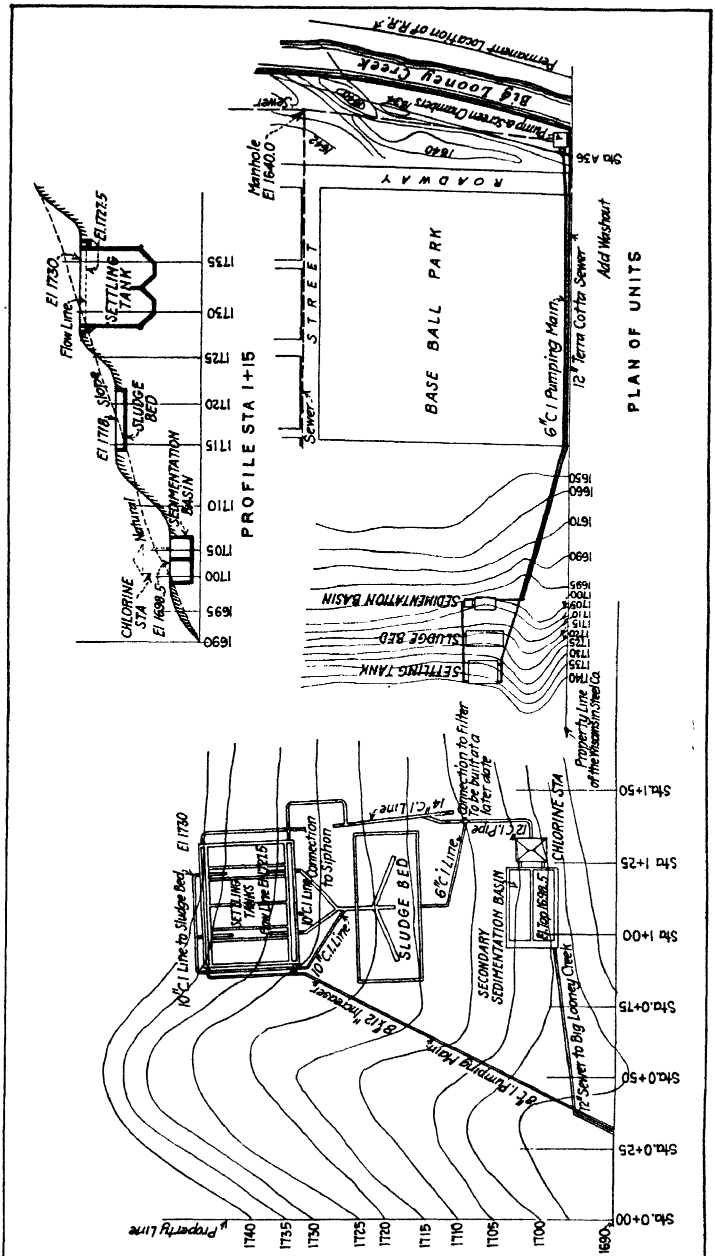


FIG. 8.—SEWAGE DISPOSAL PLANT.

as it is felt that for the present this treatment is unnecessary. The sewage treatment plant was designed by L. D. Tracy of Pittsburgh, Pa. The pipe lines were designed for a flow of 250 gal. per house per day, with allowance for 300 additional houses and for an infiltration of 40,000 gal. per mile of sewer per day.

Storm-water Drainage

The surface drainage from the various ravines extending through the town site is taken care of by open masonry culverts, the sizes of which are calculated by the formula

$$\text{Area in sq. ft.} = \sqrt[3]{\text{Area in acres}^2}$$

which gives results suitable for this mountainous country, with its steep slopes and quick run-off. The open culverts are much more easily cleaned and taken care of than closed ones.

Amusement Facilities

As the town is entirely isolated from any sources of amusement it was necessary to provide everything required to keep the inhabitants amused. A ball park was laid out at the lower end of town, and during the season the ball team furnished one of the main attractions. The ground is also made for carnivals, circuses, and such attractions; an additional park may some time be built above the upper end of town, on space reserved for the purpose.

In the lower end of the town, a large frame building for community purposes has been erected and is always well patronized. On the ground floor, Fig. 9, are located a restaurant, where both white and colored are served, in separate rooms, from one kitchen, and a movie theater; on the second floor is the gallery of the theater, which is used by colored people, the ground floor being reserved for the whites; and a room containing four bowling alleys, two pool and one billiard table, and a barber shop; on the third floor is a large dance hall with the necessary retiring rooms, and a lodge hall for the fraternal organizations. The theater seats about 450, and has a stage and dressing rooms where the usual program can be varied by occasional vaudeville shows.

A similar but larger building has been designed for the upper part of the town, but has not yet been built. This theater will have about 900 seats. It is to be built of stone. Dances are held in the hotel dining room at short intervals.

Store

With the exception of a few small stores operated by some of the tenants in the houses, where soft drinks and a few staples are sold, the store business in the town is handled by the United Supply Co. After a

careful study of the requirements by James W. Anawalt, president, plans were drawn for a store building 160 ft. (48.8 m.) wide by 100 ft. deep (30.5 m.), three stories high with a basement the entire size of the building. The outside walls of the building are of native sandstone, and

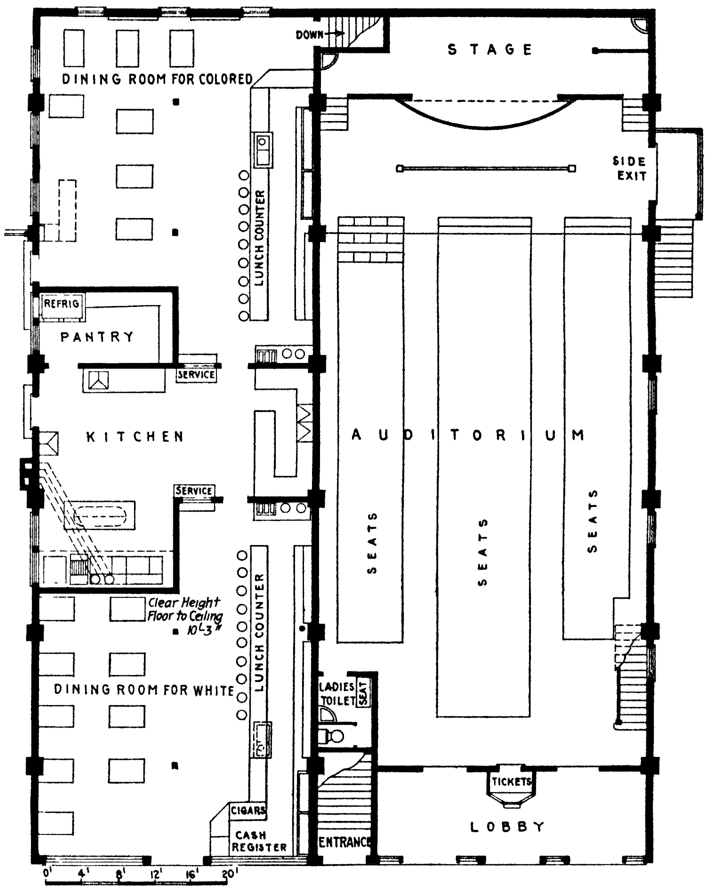


FIG. 9.—FIRST FLOOR OF COMMUNITY BUILDING.

the columns, floors, and roof beams are reinforced concrete. The roof is of structrolite blocks, which are covered by prepared asbestos roofing.

In the basement one 7 and one 5-ton machine manufacture ice, and refrigerate two cooling rooms, each with a capacity of a carload of meat, and the main refrigerator and the refrigerated counters in the meat shop in the main store room. A bakery is also located in the basement; this has a continuous type of oven with a large daily capacity of bread,

cakes, pies, etc. The basement is arranged so that a truck can be driven through it and loaded; goods kept in storage can be unloaded directly into it from cars on a siding back of the store.

On the first floor, the main store room, a drug store, meat shop, and supply room are located; and on a mezzanine are located a ladies' and children's dress and shoe department, a store room and ice-cream room for the drug store. On the floor above are one five-room and one six-room flat with bath, six bedrooms with bath for the male help, and large storage rooms for furniture, carpets, and other household goods. The main store room and the drug store are floored with linotile; the meat room is floored with white vitrified tile and wainscoted with opaque glass, and the refrigerated counters have a display of meats, butter, cheese, etc., and a large refrigerator for the meats actually being handled. The entire room is well screened and is thoroughly sanitary in every respect. So far as is known, this is the largest and most complete store for this purpose in this country and compares well with department stores in many of the cities.

Office and Wash House

The one objection to wash houses around many of the drift mines in the bituminous regions has been that usually there are several openings so that it is rather difficult to concentrate the travel to any one point where a wash house would be convenient to all. While there are several openings into each of the mines at Lynch, the main travel is through the main pit mouth in each mine; this will be the case more and more as the mines are developed. The two pit mouths are only about 400 ft. (122 m.) apart so a location midway between them is an ideal spot for the main office and the wash house with its accompanying first-aid rooms.

The ground floor of the office building, Figs. 10 and 11, is used by the superintendent, chief clerk, payroll clerks, etc., and the second floor has an office for the division engineer, a drafting room, rooms for the telephone exchange and blueprinting, a lecture room in which instruction in first aid is given, and a smoke room, which can be filled with smoke or gas in which the men are required to work with their rescue equipment in place.

It was estimated, from the experience at the Benham mine, that probably 75 per cent. of the men could be expected to use the wash room; accordingly facilities for 1500 men were provided. Experience having shown the superiority of the overhead suspension of clothes to lockers, 1500 clothes hooks were provided, each of which is separated from its neighbors by galvanized hoods, so that there can be no contact between the clothes hanging on adjacent hooks. The hangers are of the usual galvanized-iron type and are hung on galvanized-steel sash cord instead of the usual chain. The hooks can be locked in place by inserting a padlock through a loop on the cord. The number of clothes hooks determined the size of the building; the seating capacity is about 240 men.

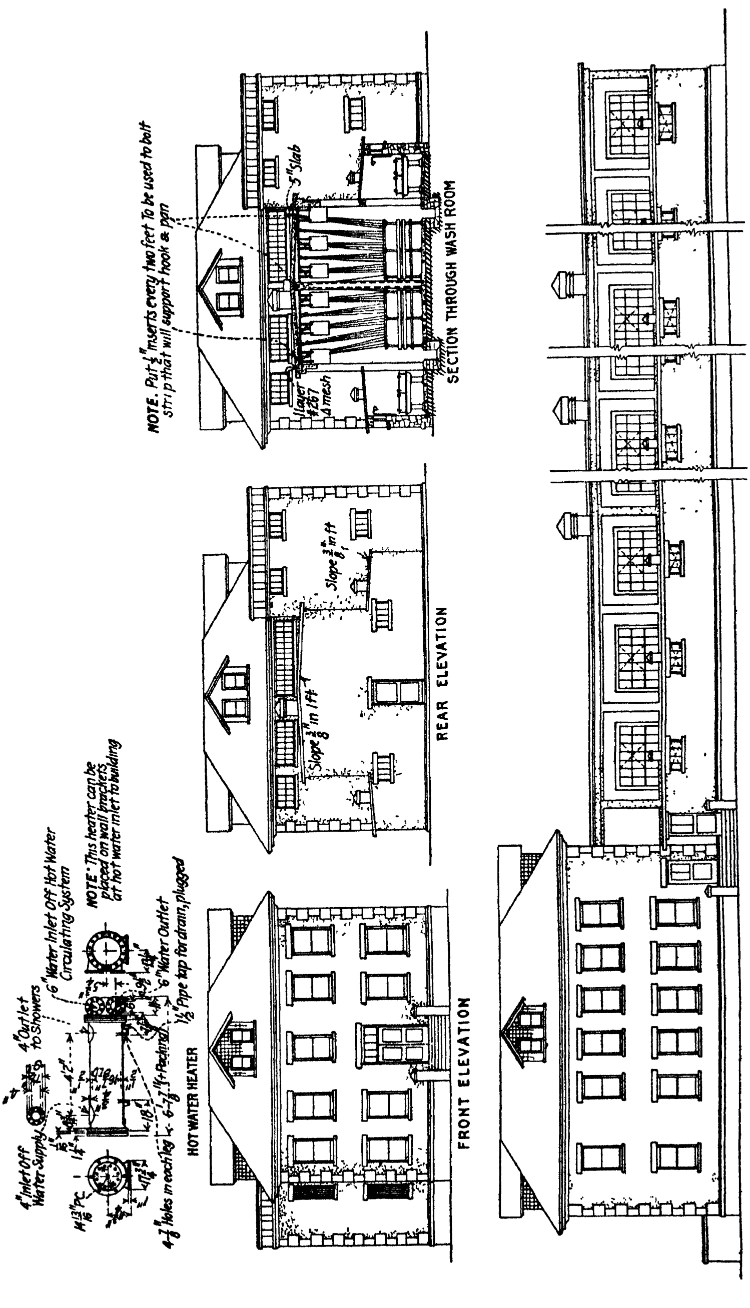
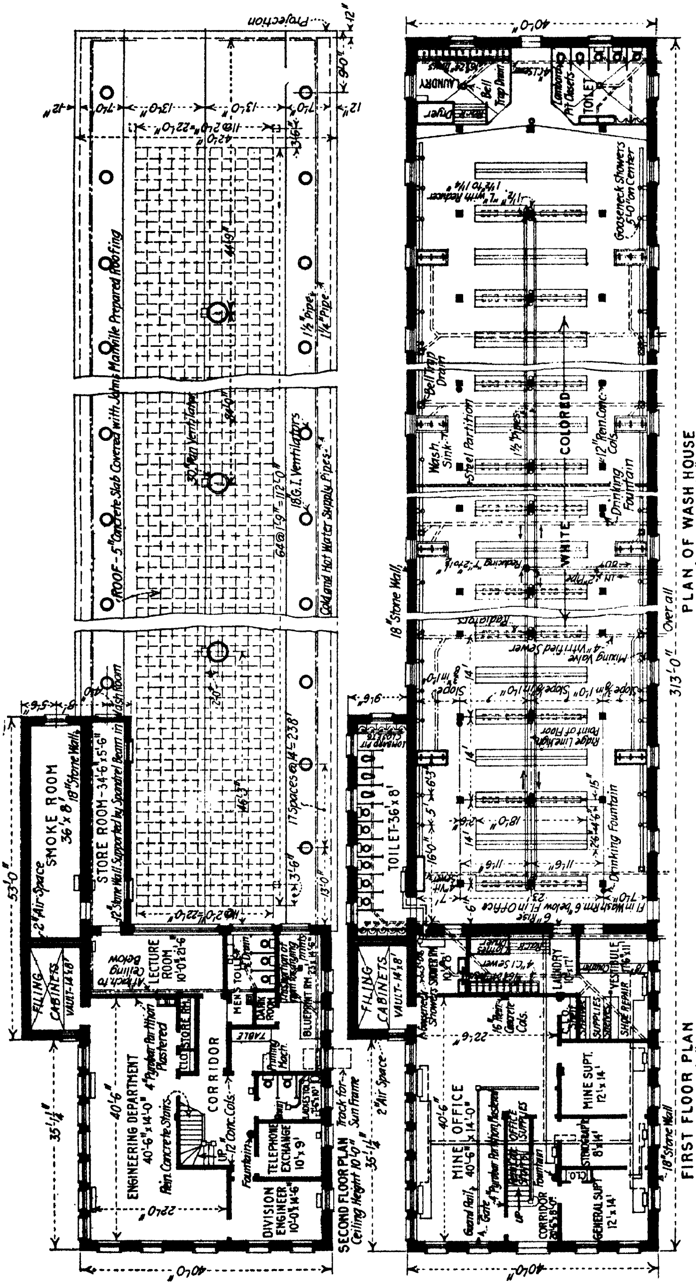


FIG. 10.—GENERAL OFFICES AND WASH HOUSE.



[FIG. 11.—PLAN OF GENERAL OFFICES AND WASH HOUSE.

As the ordinary man will not consume more than 15 to 20 min. in taking his bath and changing his clothes, the entire number can probably pass through the building in about 2 hr.; and as a man can leave for home as soon as he cleans up his place, this arrangement has worked out very satisfactorily. The building contains 71 showers and 64 lavatories. Three fans, each having a capacity of 7500 cu. ft. of air per min., installed on the main roof, have furnished satisfactory ventilation. The walls of the building are masonry and the floors of concrete; these are arranged so that all of the water will drain to one point, therefore the building can be flushed out by a hose and kept thoroughly clean.

A room is provided in which the work clothes can be washed and repaired. The building has proved to be very satisfactory and is apparently thoroughly appreciated by all who use it.

Hospital

The hospital, Fig. 12, is a stone building with reinforced-concrete columns and floors, and wooden roof. It is equipped with the usual operating room and a powerful X-ray apparatus, sterilizing apparatus of

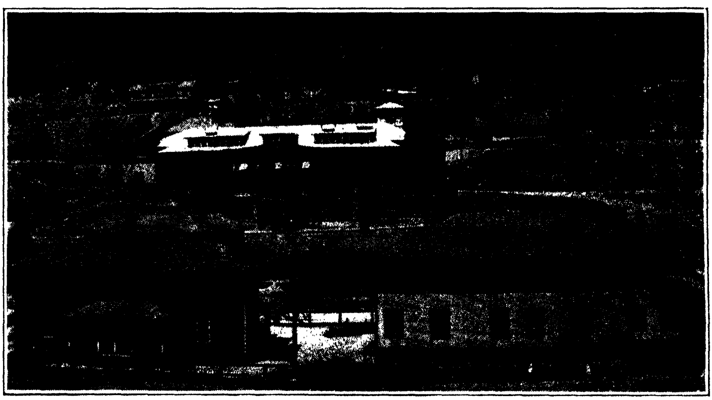


Fig. 12.—Hospital and dwellings.

all kinds, offices, and a drug room for the doctors, public wards for both white and colored patients, with a capacity of 54 beds, and a number of private rooms. The kitchen is intended only for the preparation of food for the patients, as the nurses and doctors can be well cared for in the neighboring hotel. The third floor is fitted up with bedrooms for the servants.

Power Plant

It was realized that, to obtain prompt development, electric power was absolutely necessary and three second-hand 150-kw. engine-driven, direct-current generators were installed with the necessary boilers just

as soon as the materials could be assembled; later a fourth unit was added, although ordinarily only three were operated. This temporary power plant was replaced by the permanent plant in August, 1919.

The only public utility company within reaching distance of this plant had two small power stations, but a substation at Lynch would have had to depend on a single transmission line 8 mi. (12.9 km.) long over the top of Big Black Mountain, which it would cross at one of the highest points in the state. This line would be liable to considerable trouble, and would be comparatively inaccessible for repairs. It would probably be years before a loop line could be constructed and as the power company required the coal company to finance the necessary extensions, the latter company decided to install its own plant in order to have a reliable source of power at all times.

A careful study showed that alternating current to inside substations, for the mine transmission, would result in very large savings. On account of the distance that would ultimately have to be reached and of the possibility of the acquisition of property 3 or 4 mi. west of the power plant (which property has since been acquired) it was decided to generate at 6600 volts. The new plant accordingly was laid out for two 1875-kv-a., three-phase, 60-cycle, 6600-volt, turbo-generators with the necessary switchboard and control apparatus. Each machine is mounted on a heavy steel framework, under which is placed the surface condenser; directly under the condenser is placed the radojet air pump, and the centrifugal circulating pump. The arrangement is compact and accessible. The foundations of the pumps and condensers are built on the ground floor and the floor between the turbo-generators is covered with an open steel grating, which allows the air to ascend from the condenser floor and also permits the use of the crane for any repairs to the condensers, as the building is entirely open after the gratings are removed. The general layout of the plant is shown in Fig. 13. The cold well is under the floor of the main turbine room and the circulating pumps discharge into a steel-pipe line, which feeds the water into a spray pond built as a part of Looney Creek, which is outside of the power plant. Sufficient room has been left in the power plant to install an additional turbine as large as 10,000 kv.-a., should the demand warrant it, and the crane is large enough to handle a machine of this size.

The switchboard is on the same elevation as the turbo-generators and the control apparatus is in a room immediately back of it. On top of this room are the lightning arresters and the disconnecting switches on the outgoing lines; in the room immediately below it are the necessary current and power transformers for the switchboard instruments and the 6600–440 volt power transformers for the fan and tipple motors.

Adjoining the turbine room is the pump room, in which are installed two centrifugal boiler-feed pumps, each with a capacity of 600 gal. per

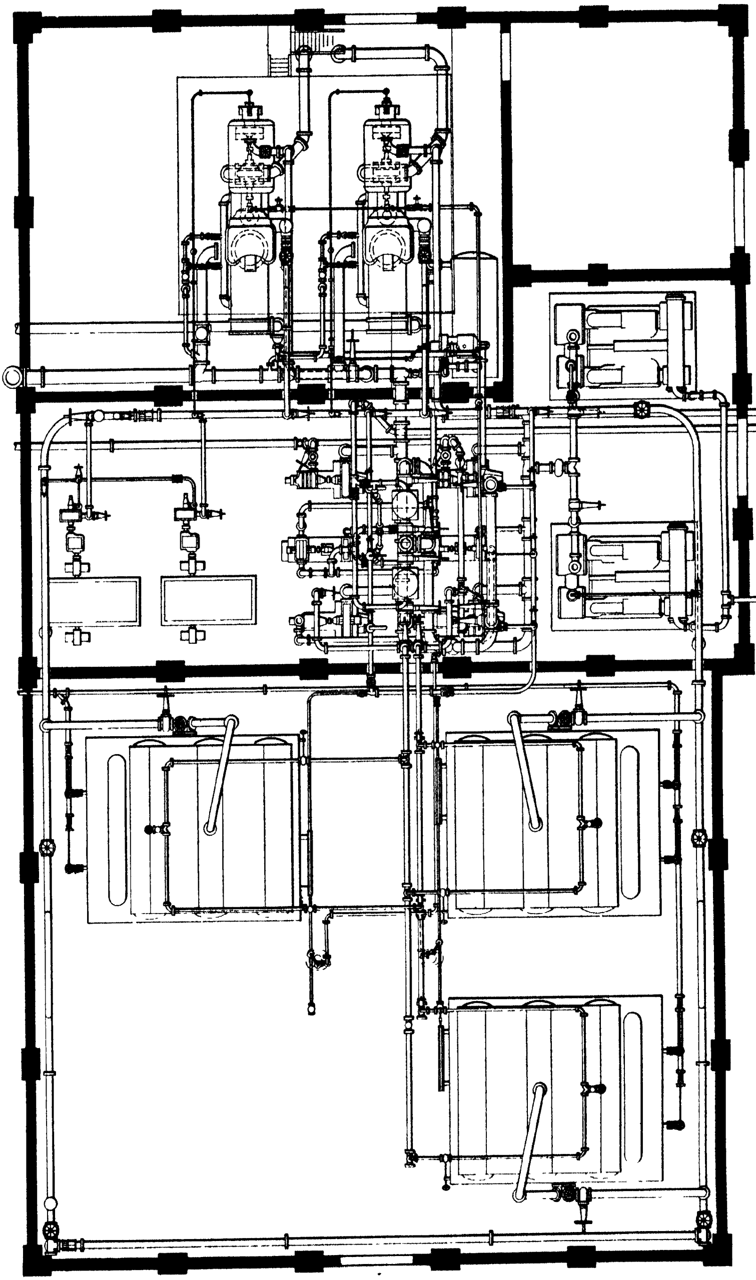


FIG. 13.—GENERAL PIPING LAYOUT LYNCH MINE POWER STATION.

min., two centrifugal pumps for the town water supply, each with a capacity of 800 gal. per min. against a 500-ft. (152 m.) head, two centrifugal pumps for circulating the hot water through the central heating system, and two fans for the forced draft for the underfeed stokers. All of these machines are driven by steam turbines, the pumps by direct connection and the fans through gear reductions. Two steam engine-driven compressors supply air for pumping the wells, operating the rotary dumps in the tipple, greasing mine cars, operating the loading gates under the bin, cleaning motors, and shop tools, etc. These machines have a combined capacity of 2200 cu. ft. free air per minute to 110 lb. pressure. Immediately adjoining the pump room is the boiler room in which are installed three 750-hp. Sterling water-tube boilers, operating at 175 lb. steam pressure and 100° superheat. There is ample room inside the building for a fourth unit and provision has been made outside for the installation of three similar units should it become necessary. These boilers are equipped with Taylor underfeed stokers, an underground ash-cleaning system, and an overhead steel coal bin with a capacity of about 250 tons, from which the coal flows by gravity to the stoker hoppers. This overhead bin is filled with slack coal and any mine refuse delivered by a belt conveyor from the tipple. The boilers are connected by a steel breeching to a reinforced-concrete chimney 9 ft. (2.7 m.) internal diameter and 205 ft. 62.5 m.) high.

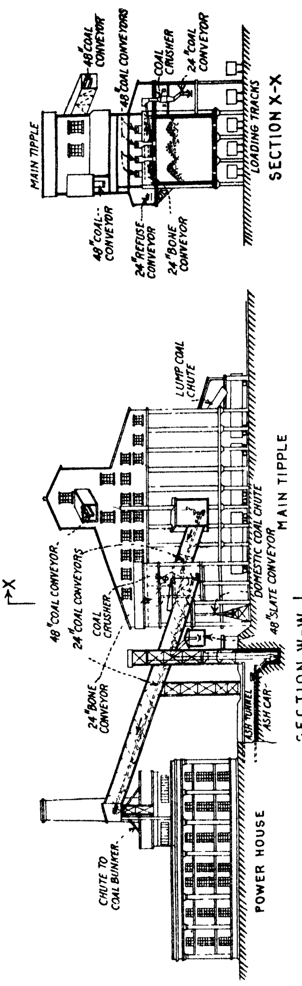


FIG. 14.—SECTIONAL VIEWS OF TIPPLE.

The ashes are taken from a small storage pit under the back of the boiler settings, by roller-bearing cars, to a point outside the building, where they are dumped into a skip hoist which delivers them to the slate conveyor, Fig. 14, which carries them to a bin on the mountain side. The skip hoist is also arranged so that any garbage or refuse of any kind

collected around the town can be dumped into it and disposed of with the ashes and the mine waste.

The steam lines are all of extra heavy steel pipe, flanged, with Van-Stone joints, built in a loop entirely around the plant, and provided with the necessary expansion joints, so that any unit can be operated from any one of the boilers. The entire building is made of reinforced concrete, sash and doors are all of steel, floors are of concrete with steel-plate coverings over the pipe trenches; no wood is used anywhere in the building.

For a considerable distance above the power plant, masonry walls have been built along both sides of Looney Creek, to confine it within a definite channel, and part of this area has been utilized for a spray pond, the water being kept in it by a movable steel dam at the lower end of the power plant. When there is ample water in the creek, the condensing water is drawn directly from it through movable screens, to keep out leaves etc.; when the supply is scarce, the dam is raised and the spray pipes put in operation.

Tipple

As stated before, the output for which the plant was designed is 2,500,000 tons per year, or roughly, 8000 tons per working day; and as it was expected that this output would eventually be increased, and because any tipple should be designed to handle easily the coal at the greatest rate at which it can be delivered to it, which is usually during one of the morning hours, the tipple was planned for a mine capacity from each side of 1000 tons per hour, or a total capacity of 2000 tons per hour, or 16,000 tons per 8-hr. day. As all of the coal was to be shipped for coking purposes, no preparation, so far as sizing was concerned, was necessary; but on account of the probability of working more than one seam to the tipple and also on account of the likelihood of encountering occasional impurities in the seam, it was decided to equip the tipple with picking tables so that the coal could be cleaned by hand picking. The experience at the Pocahontas mines had shown that it is not feasible commercially to pick coal less than 1¾ in. (4.45 cm.) in size, so the screens were designed to make coal over 6 in. (15 cm.), between 6 in. and 3 in., and between 3 in. and 1¾ in.; all the coal under 1¾ in. goes directly to the bin. The picking tables were installed with a capacity of handling the output of each size at a speed of 60 ft. (18 m.) per min. and at a depth on the table of not more than the size handled, or in other words, in the size from 1¾ to 3 in. the coal would completely cover the picking table but not more than 3 in. deep. Two tables were installed for each of the three sizes, making a total of six picking tables.

The location of the plant, at the end of a single-track railroad, on a 2 per cent. grade and about 5 mi. from the nearest railroad yard, made necessary the use of a number of trains to keep it supplied with cars, so

the track system, both above and below the tipple, was laid out for a capacity of 240 cars, or 12,000 tons per day, thus allowing for some margin over the day's run and also for a future increase in capacity. The empty tracks were designed for 60-car trains and the incoming train is pulled through to the upper end of the storage yard and dropped into the tipple tracks. Below the tipple, the cars are dropped over a railroad scale and weighed and then into the storage yard with tracks of sufficient length for 90-car trains. All of the tracks are laid with 80-lb. steel rails.

Even with this provision, it was felt that there would be many times when the plant operation would be hindered by the shortage of cars and that many times, while cars would be expected during the morning, the men would not enter the mine because they could not see them and for that reason on many days the run would be stopped because the cars on hand had been loaded. It was, therefore, thought advisable that a 5000-ton storage bin be provided.

The coal is of a hard splinty character and comes from the mine in large slabs. As coal passing over the 6-in. (15 cm.) screen was likely to be entirely too large to load through a hopper, the tipple was designed to deliver this large coal to the lower end of the bin where it is loaded separately by chutes into cars. After a careful study of the most economical shape of bin for the desired capacity and the space necessary for the picking tables, screens etc., three loading tracks were decided upon, the bin being placed directly above these. The coal is loaded from the bin into the cars through openings approximately about 2.5 by 3 ft. (0.7 by 0.9 m.) which are closed by steel plates, operated by air cylinders.

The use of a bin meant that the picking tables would be a considerable distance above the ground; and as it was advisable, on account of reducing the average haul of the coal and for the purpose of draining as much of the coal as possible, that the pit mouth be placed as close to the flood level as was practicable, conveyors were installed for raising the coal from the ground to the top of the tipple.

On account of the extremely large capacity desired, after the inspection of several plants, rotary grizzly screens were chosen for the smaller sizes and bar screens for the larger sizes. Shaking screens would have required very large units, which would have meant considerable vibration. The rotary grizzly screens revolve at slow speed, consume little power, and screen efficiently.

The use of steel-pan conveyors was carefully considered and plans and specifications for a steel tipple with such equipment and shaking screens were prepared. After the inspection of other plants, however, it was decided to use rubber belt conveyors, the only objection to their use being that the tipple was removed farther from the dump. Even with the increased use of steel involved, however, these conveyors were much cheaper than the steel-pan conveyors would have been and

their size was well within good practice. Roller-bearing idlers of heavy design were specified throughout and the top and bottom pulleys and the gear drives are of unusually heavy and rugged design, all gears being of cast steel. The use of as few sizes of shafts as possible throughout the tipple was specified, even if some were a little large, in order to reduce the number of boxes and repair parts to be carried in stock.

Around the usual coal mine, the disposal of the slate and mine refuse, if any is encountered, is a matter of great difficulty; in the writer's opinion, in most mines, not enough attention has been paid in the design of the plant to handling this material, which is therefore unnecessarily expensive. On account of the thinness of the coal in some places and as draw slate would probably be encountered in some headings, provision was made for handling 400 tons per hour through the tipple and to the slate dump, by the ordinary tipple crew. As the cars used were of the solid-end type, the use of rotary dumps was necessary; and so in order to get the capacity desired, it would be necessary to dump two cars at once, therefore the equipment was designed so that two cars of coal, two cars of slate, or a car of slate and one of coal could be dumped at the same time. The handling of such a volume of coal and slate at each dump meant the use of separate belt for handling the slate.

The rotary dump, Fig. 15, discharges the contents of the cars into hoppers under the dump, these being so arranged and so covered by hinged gates that coal falls into one bin and slate into another and, by air-operated valves, the contents of any car can be easily diverted into either bin. From these bins, the coal passes to the coal conveyor, and the slate to the slate conveyor, by short apron feeders of very heavy rubber belts, Fig. 16. The two main conveyors are carried in the same galleries to the tipple, the coal conveyors in each case being over the slate conveyors.

Rotary dumps are ultimately to be installed in duplicate in each dump house but at present one of them has only one dump. The dumps are operated by compressed-air cylinders and revolve 135° and back to the starting point, there being a positive stop so that the rails in the dump come exactly to the same elevation as those leading to the dump, at each revolution. The dumps are guaranteed to have a capacity of four dumps per minute, or, as the cars hold about 3 net tons, a capacity of 24 net tons per minute. This can easily be maintained over long periods of time and, for short periods can be exceeded.

For each mine, storage tracks above the dumps hold about 100 mine cars, which are automatically fed into the dumps by car feeders. After leaving the dumps, the empty cars are handled by trip makers, which are also used to raise the cars to a higher level, in order to save excavation. The height from the dump to the conveyor feeder is regulated entirely by the arrangement of chutes and, as this was designed so that either car

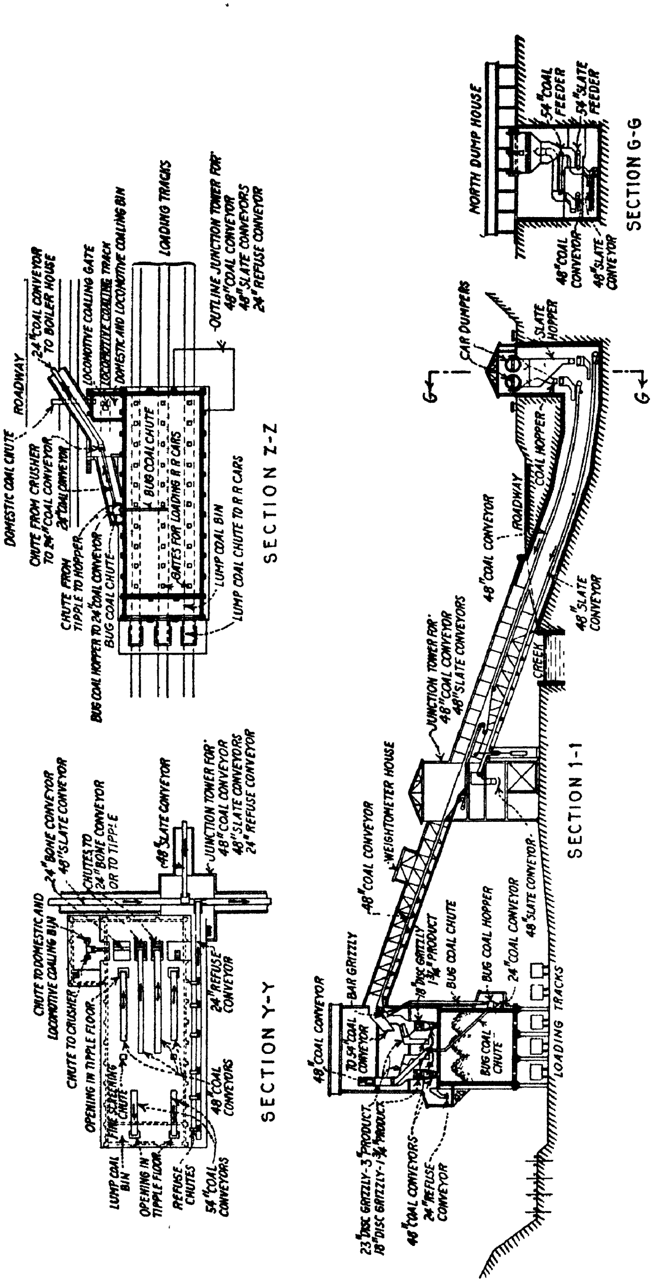


FIG. 15.—SECTIONAL VIEWS OF TIPPLE

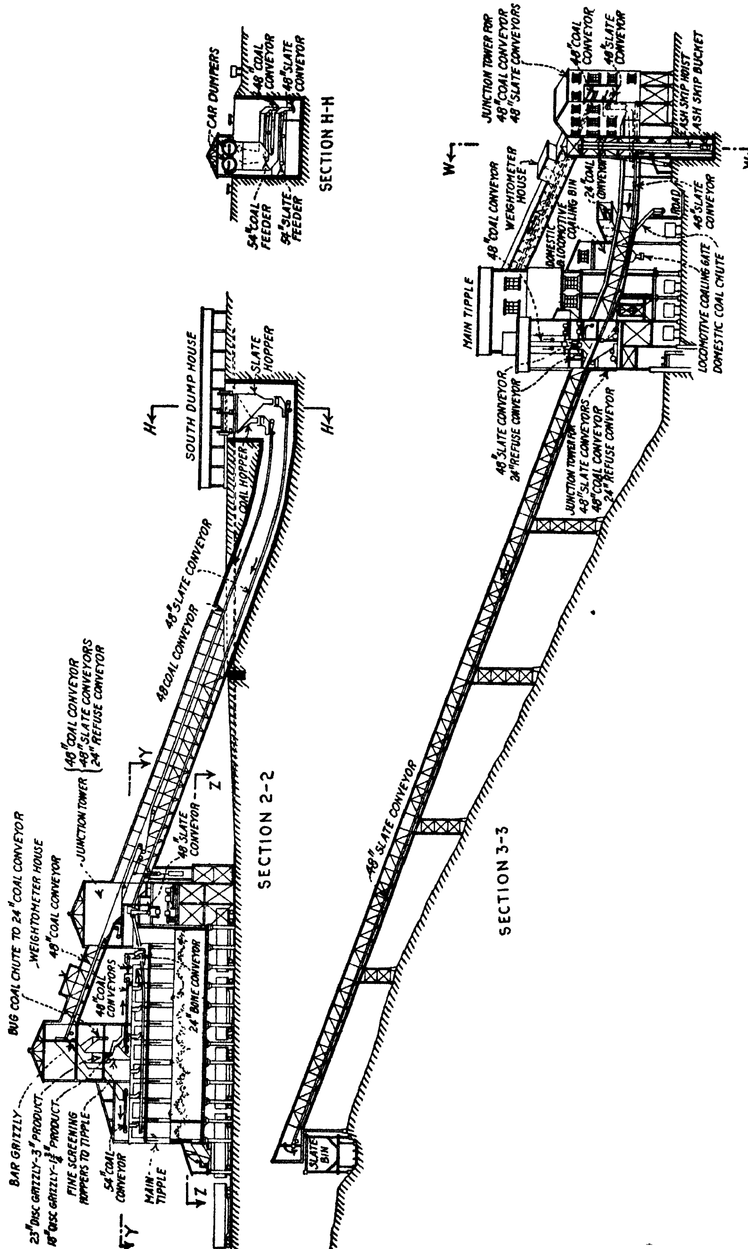


FIG. 16.—SECTIONAL VIEWS OF TIPPLE.

could go into either dump, it required a considerable excavation below the dumps, the deeper of the two conveyor pits being about 54 ft. (16 m.). The conveyor pits, as well as the lining of the conveyor galleries from the dump pits to the surface of the ground, are made of reinforced concrete, the outside of which in each case was water-proofed with tar and felt paper. The supports for the floors in the dump hoppers are of heavy steel girders.

From the dump pit in each case, the coal conveyors carry the coal to the top of the tipple, where it passes over a bar grizzly having a 6-in. opening, Fig. 15. The coal passing over the bar grizzly is carried by chutes to the lump-picking tables and is then dumped into the lump coal bin at the end of the tipple.

The coal passing through the bar grizzly goes over a 23-in. (58 cm.) revolving disk screen, Fig. 15, which takes out all material larger than 3 in., this material goes through to the picking tables and then to the main bin or through a chute to the domestic- and locomotive-coal bin. The material passing through this screen goes to an 18-in. rotary disk screen, which separates the material larger than 1¾ in.; this goes through picking tables, thence into the bin.

In the morning, the first coal loaded out of the mine is the "bug dust," or machine cuttings. In certain parts of the mine, a small parting about 3 in. above the bottom, which occasionally is 1 in. thick, and is considerably higher in ash than the rest of the coal is cut out by the machines. Occasionally, also, the machines dig into the fireclay bottom, which is mixed with the bug dust. As this bug dust is loaded into separate cars, it is easily seen on the conveyors; so by a flap gate at the end of each conveyor it is sent direct to a bug-dust hopper, Fig. 15, from which conveyors take it to the boiler house.

The material from the picking tables, if of a combustible nature, is dropped to the floor and, at suitable intervals when the work is slack, is deposited on the conveyor under the picking-table floor, which dumps it into a crusher in which it is reduced to a nut-coal size; here it drops to a conveyor and goes to the boiler-house coal bin. The 3- to 6-in. (7.6 to 15 cm.) lump coal can be handled this same way, if desired. The other material from the picking tables is wheeled to the side of the bin and dropped through suitable refuse chutes to the belt conveyor which deposits it on a slate conveyor. The locomotive coaling bin is used for coaling locomotives on the railroad, which is done through a gate where the coal is weighed; the coal from this same bin also drops by a chute to a point along the road, whence it can be taken to the houses by truck or wagon.

The slate from the north-side dump, Fig. 17, is taken by a slate conveyor to the junction tower where it is dumped upon a conveyor, Fig. 16, which finally deposits it in the slate bin on the side of the

mountain. The slate from the south-side dump is taken on a conveyor to the junction tower at the end of the bin, where it is also placed on a slate conveyor for final disposition. The slate bin on the mountain side has a capacity of about 400 tons and the slate is drawn from it through air-operated bin slides into a steel stacking larry, which is so arranged that the slate is drawn from the bottom of the larry by a conveyor that can discharge to either side or in front, thus doing away with the necessity of moving tracks at frequent intervals. This larry can build its own track directly ahead of it or it can make a fill from 30 to 40 ft. (9 to 12 m.) wide on top. Pieces of slate weighing as much as 200 lb. are handled by this larry and are discharged at a distance of from 15 to 20 ft. from the center of the track.

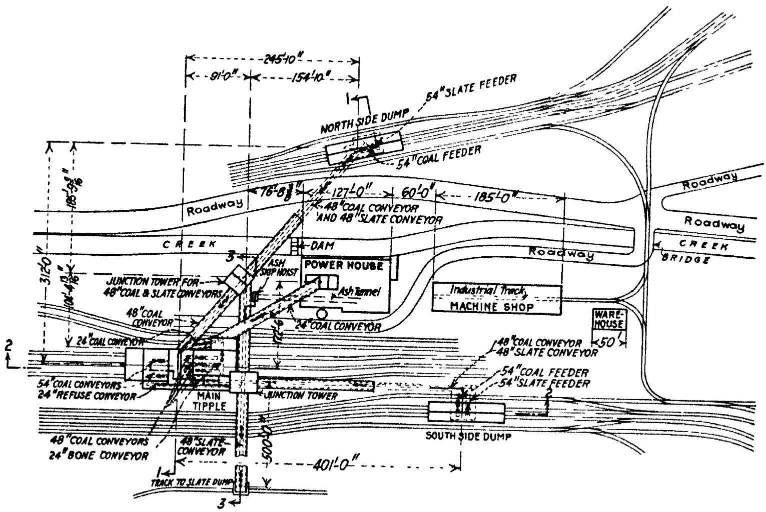


FIG. 17.—GENERAL ARRANGEMENT OF PLANT.

The coal is drawn from the main bin through cast-iron boxes, which are embedded in the concrete floor and are closed by heavy steel plates, sliding in cast-iron grooves. These gates are operated by air cylinders having a stroke equal to the opening of the gate and are about 30 by 36 in. in size. They are spaced approximately 12 ft. apart and the cylinders are operated by three-way cocks, which are turned by the loader walking along the platform, from which he can estimate the proper loading of the cars. Three cars can be loaded on each track, or a total of nine cars, at one time and it is possible, if the bin is kept full of coal, to load 500 tons of coal in 3 min. The lump coal at the end of the bin is loaded through chutes, which are raised or lowered, to shut off the coal supply, by electric windlasses, Figs. 15 and 16. The main bin is

built of reinforced concrete and was calculated to have a working stress of 500 lb. per sq. in. for the concrete and 16,000 lb. per sq. in. for the steel. So far as the writer has been able to ascertain, it is the largest reinforced-concrete bin in the world. The superstructure on top of the bin, the conveyor galleries, and the junction towers are of steel covered by corrugated galvanized sheets. An idea of the size of the tipple may perhaps be obtained from the fact that there are 1100 tons of structural steel in these structures and in the dump pits.

As it was realized that considerable dust would be forced through the openings in the picking-table floor, by the coal dropped into the bin below, four large steel ventilators extending from the top of the bin through the top of the building were built. It was not thought that the coal would be dusty enough to cause any inconvenience in the rest of the structure, but it has been found that both at the dump pits and in the tipple house the dust is a serious problem, so a ventilating system is being provided. All machinery is electrically driven, using 440-volt induction motors and spurgear speed reducers.

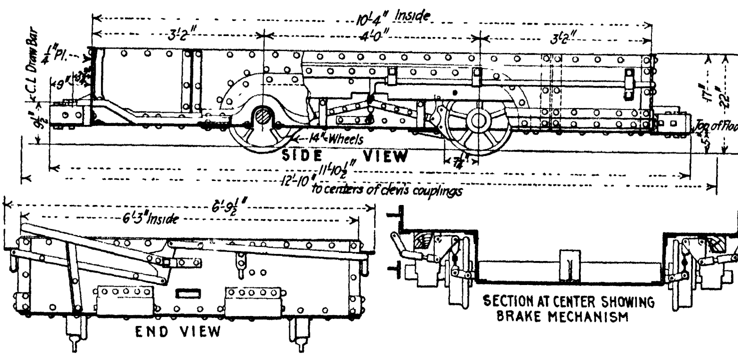


FIG. 18.—STEEL MINE CAR, MODEL C.

MINE CARS

The track gage in all the company's mines in the Pocahontas field is 48 in. (1.2 m.); and as this has been very satisfactory there was no reason to change it, in the Lynch mine, especially as some of the mine cars from the Gary mines were to be used there until the permanent equipment was installed. A great deal of thought was given to the mine car to be used, as it was desired that this should have a capacity of at least 3 net tons and should be as low as possible. Experience has shown that a solid-end car is much more serviceable and has much less cost of upkeep than an end-gate car, for which reason rotary dumps were installed. Some years previous, in some mines in which several of the officials had been interested, a steel car having the bottom of the car under the axles

had been fairly satisfactory. The weak points in this design were studied and the style of the car changed to adapt it for roller bearings, which were placed in self-aligning cast-steel boxes. The car shown in Fig. 18 is the result. This car stands about 22 in. (0.55 m.) above top of the rail and the motion required to throw the coal into the car is sufficient to raise it over the side, so that no actual lifting of the coal beyond this is necessary. The wheel base was made 48 in. in order to prevent the long overhang usual in mine cars and to insure a steadier running car, which has been the result. When first put on the track, as is usually the case with steel cars, the cars were quite stiff and gave considerable trouble by jumping the track, but after the first week's run, this trouble disappeared and the cars have been very satisfactory. The first axles were made of nickel steel with a hardness of 250 Brinnell, but the later ones were made of open-hearth steel forgings, heat treated to give the same hardness. The wheels are pressed on the axle; cast-iron, rolled-steel and cast-steel wheels have been used, depending on the material available at various times during the war. Both the cast-steel and the rolled-steel wheels have been very satisfactory.

The boxes are greased every 6 mo., a slow semi-fluid grease being used for this purpose. Greasing is best accomplished by connecting a compressed-air hose to the end of the grease gun. When the end of the gun has been inserted in the bearing, the air pressure is turned on, the old grease is forced out and the new put in place instantly. The bodies for the first 500 cars were purchased, the trucks being assembled at the works, but the remaining 1200 cars were built complete at the shop at Lynch.

The Shop

On account of the isolated location of the plant, ample repair facilities for all types of machinery used were absolutely imperative. The repair shop was the first building designed and built. It is of steel and glass construction, with structolite roof slabs covered by asbestos sheet roofing. It has a 5-ton traveling crane, four forges, a large bulldozer, steam hammer, cutoff saw, punch and shear, wheel press, boring mill, drill press, two lathes, and the necessary grinding machines, air hammers, tools, etc. Almost any repairs can be made to any of the machinery used and armatures for any of the motors can be rewound. Brass castings and any small brass parts for mining machines and locomotives are also made.

Fans

The two fans are duplicates, each being primarily blowing but set to be reversible, and having a capacity of 300,000 cu. ft. per min. against a mine resistance of 3 in. water gage. They are set at right angles to the pit mouth, and in the ducts leading to the mines explosion doors are placed directly in front of the pit mouths.

The fans are driven by slip-ring, 440-volt, alternating-current motors through spur-gear speed reducers. These reducers are large enough to develop the full capacity of the fan, but the motors are smaller and of lower speed than will ultimately be required. When the mine requires larger motors, the present ones will be used as spares for the tipple. As the ratio of the speed reducer is fixed the fan will be driven faster. The fan settings and buildings are, of course, entirely fireproof, being of concrete and steel.

Central Heating System

On account of their proximity to the power plant, the main buildings, and many of the better grade houses are heated by a central hot-water heating plant. While the first cost of the central plant is about 15 per cent. more than that of the separate plants, the operating expense is much less and it was decided to install a hot-water system.

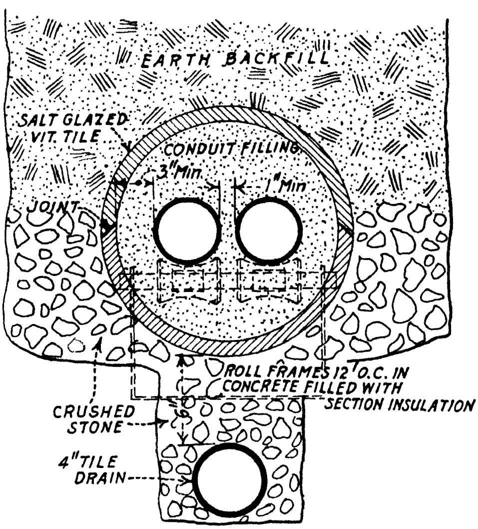


Fig. 19.—Conduit for central station hot-water heating system.

The highest building to be heated is about 164 ft. above the power plant; so to avoid pumping all of the water against this head, the system was made an entirely closed one. The cool water returning balances the outgoing hot water and the head against the pumps is only that due to the friction loss in the pipes. The water is heated in a closed heater by exhaust steam; the usual temperature is 160°, but it can be heated to 205° in periods of extremely cold weather. The makeup water is taken from the condenser circulation and is passed through a deactivator, to remove its oxygen content. This machine consists of a tank filled with loose thin

steel sheets, upon which the oxygen acts, and a sand filter to remove any foreign material from the water. The deactivating plant was designed by F. N. Speller, and experience with similar plants in hot-water heating systems in large apartment houses has shown that the corrosion in the pipes has been largely eliminated.

All the pipe lines are of steel, flange joints being used at frequent intervals, and expansion joints at all branches and elsewhere when necessary. Both main and return lines are laid in the same trench and insulated by surrounding them with asbestos felt, which is held in place by vitrified sewer pipes, the sizes of these depending on the sizes of the water lines, Fig. 19. The sewer pipes were placed in trenches filled with crushed stone to about the middle of the pipe. In the bottom of the large trench is a small trench in which a 4-in. sewer pipe is placed for drainage. The expansion joints, bends, etc. are placed on small concrete pedestals.

Each building is provided with valves on each line, so that the velocity of the water can be regulated; provision is also made for taking the supply of hot water from the main, so that a constant supply of hot water is available at all times. In the summer time this is provided without operating the pump by connecting the main to the tank, which furnishes the necessary pressure.

When the lines were first laid, considerable trouble was caused by the ground water encountered, the 4-in. pipe being too small to pass all of it. Arrangements were made for additional taps from this line at various points, which relieved the trouble. A larger line would be better in ground of this character.

The Mines

To develop the mines quickly and allow the production of coal while the permanent plant was under construction, headings *A*, *B*, *C*, *D*, *E*, *G*, *K*, *L*, *M*, *N*, *O* and *P*, Fig. 21, were started from the outcrop and temporary tram roads were built along the hillsides connecting them; allowing the shipment of coal from temporary tipples near *A*, *D*, and *M* headings. As the capacity grew, a large temporary wooden tipple was built across the hollow just below the site of the permanent one, and gradually the shipments, with the exception of the coal from *A* and *B* headings, was concentrated here. Before the permanent tipple was completed, shipments over this temporary one exceeded 100,000 tons per month.

The permanent pit mouths were the last ones started and in each case a slide was encountered, which necessitated some heavy steam-shovel excavation. Coal from each one, however, was shipped over the large temporary tipple and before the permanent one was complete the haulage roads were completed so that all of the coal from all places could reach the two main pit mouths.

The mines are laid out on the advancing robbing system, each unit of four-room headings forming a separate section, in which the upper two-room sections are worked out and the ribs drawn as the section advances. The third room heading is really a set of main headings, there being an empty and a loaded track and two airways, the room sections on each side form barrier pillars while the section advances and are drawn retreating, when the boundary has been reached. Some of these headings will be 3 mi. (9.6 kg.) long and with this system practically a uniform output can be maintained from a section until it is nearly ready to be abandoned. [Each section will give at least 1000 tons per day, and five of them are intended to be under development in each mine at the same time, to maintain the desired output and to be ready for a reasonable increase.

Headings are driven about 12 ft. (3.7 m.) wide, with ample clearance on each side of the car for safety, and on 60 ft. (18.3 m.) centers. Airways are usually 18 to 20 ft. wide. Rooms are usually 36 ft. wide and their distance apart varies from 80 ft., under light cover, to 120 ft. under the main mountain. It will be noticed, from Fig. 21, that mining is now being done under 1500 ft. of cover.

Cutting is done on the bottom by shortwall machines. Gathering is done partly by ponies and, where the coal is too low for this, by storage-battery locomotive. In

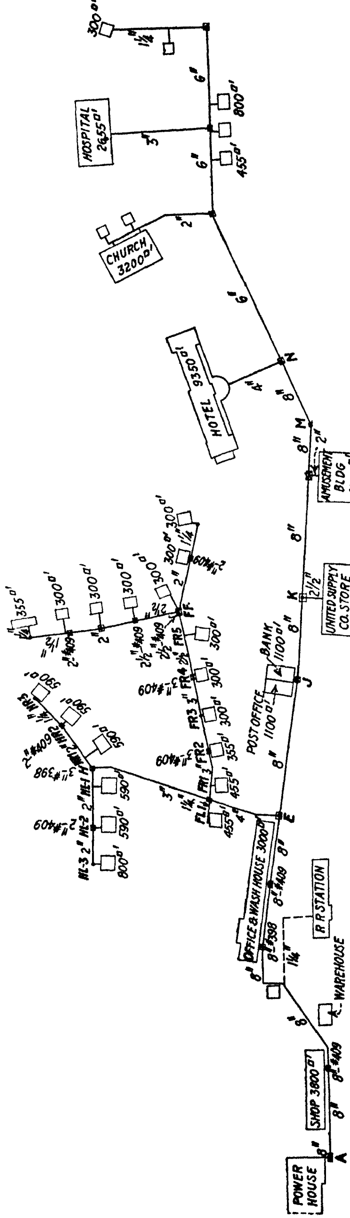


FIG. 20.—CONDUIT LAYOUT FOR CENTRAL STATION HOT-WATER HEATING SYSTEM.

the early workings, many local dips were encountered and a large number of electric room hoists are used to pull the cars from such places.

There are four inside substations, each of 200-kw. capacity, in the mines. On account of the heavy cover and the expense of bore holes from the surface to the mines and of the difficult and expensive pole-line construction, it was found cheaper to transmit the current in the mine by cables. The size of wires depends, of course, on the load and distance, and they are installed as three conductor cables with heavy rubber and

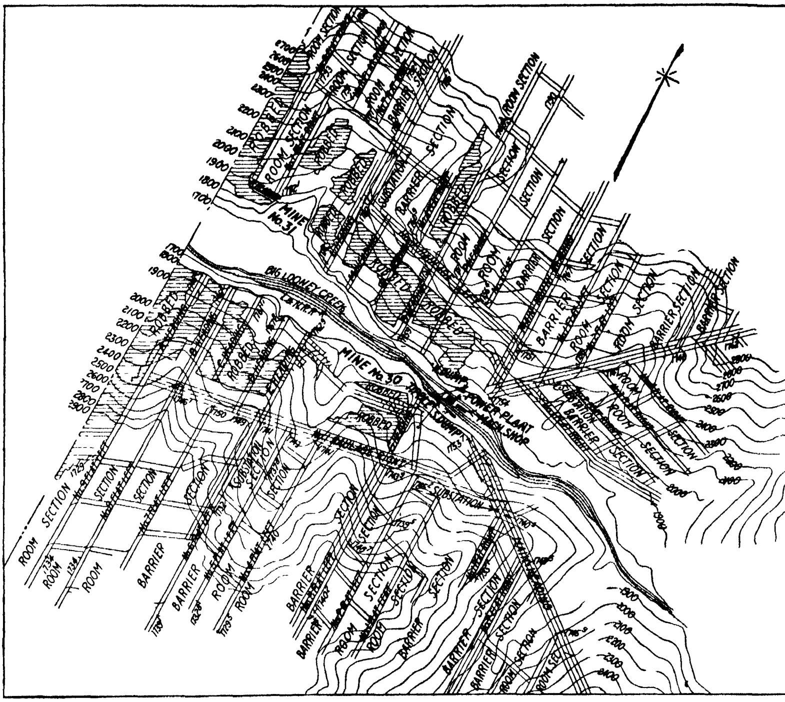


FIG. 21.—MAP OF MINE WORKINGS.

braid insulation, designed for a working pressure of 13,000 volts. These cables are placed in 3 or 4-in. fiber conduits, which are laid along the side of the heading and surrounded by concrete with a minimum thickness of 3 in. (7.6 cm.). The telephone lines are laid in a smaller conduit, placed near the power line, and embedded in the same block of concrete. The substation rooms are of fireproof construction (of reinforced concrete and steel) and are placed at the side of a main heading and with a connection to an air course, so that a circulation of air can be maintained through them. They are all equipped with automatic reclosing circuit

breakers. The direct current is transmitted from the rotary converters by 500,000 cir. m. bare feeders and 0000 trolley wires, the feed cables being of such length that the voltage will not drop more than 50 volts at full load on the locomotives operating in that section.

All haulage headings have at least 5 ft. clearance from top of the rail. Main headings are laid with 60-lb. rails, room headings with 40-lb.; 20-lb. rails are used in the rooms. Both rails of all haulage tracks have electrically welded joints to make good returns.

No separate manways are provided as the haulage roads have at least 3 ft. clearance between the side of the heading and side of the cars, have ample headroom, and are lighted by 40-watt lamps at intervals not exceeding 125 ft.

Construction

During the time this plant was under construction, labor conditions were probably worse than they had ever been in this country, and to make matters worse in this immediate locality, the large plants at Muscle Shoals, Ala., Nashville, Tenn., and the camps at Louisville, Ky., all within a few hundred miles of Lynch, were under construction. It was impossible to make contracts at any fixed prices, and as there was absolutely nothing at the site and everything had to be brought in, it was decided to do all work by company forces, with the exception of a portion of the early railroad grading, the installation of the asbestos roofing on the permanent buildings, and the placing of the insulation on the heating system piping, which were done by contract.

In addition to a large number of shanties required, a number of large mess rooms were built, which were all operated by the company, and to the excellence of the meals served in these rooms is largely due the success achieved in securing the amount of labor required. The entire construction camp was carefully policed, the water supply carefully watched and the health of the camp was excellent, not a single case of typhoid fever developing until the construction period was over.

Due to the fact that only about ½ mi. of track had to be built to enable shipments to begin, the first coal was shipped in 63 days after work was started, on Nov. 2, 1917. Since then the output has been as follows: 1917, 12,400 tons; 1918, 541,000 tons; 1919, 1,243,000 tons; 1920, 1,338,000 tons.

DISCUSSION

E. V. d'Invilliers, Philadelphia, Pa.—Will the author explain more clearly the water supply arrangements? A mine of this size would demand a very large supply.

Howard N. Eavenson.—The water supply was, and still is, a source of considerable anxiety. In very dry years, even Looney Creek is almost dry. It was thought that we could get enough water from wells

so a dozen or more were drilled but the supply was found to be too small. A pipe line was then laid from Gap Branch, tapping it above the houses, and the water was run into the tank by gravity. The next year, a pipe line was laid from Looney Creek above the houses but even that was not as good a supply as was desired. Now, the company has driven several rooms, as a reservoir, in the mine under the Gap Branch, and their mouths are dammed; it is estimated that they will hold enough water for six or eight months. A number of samples of the mine water have been tested, but it has been impossible to detect any taste or anything like bacteria or acid. There were only two other alternatives: one was to build a pipe line from the mouth of Looney Creek, about 4 mi. and the other was to drive a heading under Looney Ridge and then bring water through the mine. Either would have proved expensive, but it is thought that the present supply will be entirely adequate.

Cadwallader Evans, Jr., Scranton, Pa.—What is the probable life of the property that it will justify such permanent improvement?

Howard N. Eavenson.—There are about 250,000,000 tons of coal that can be worked with that plant. There are four seams above water level and one below. The whole 19,000 acres can be worked through this plant, and as its capacity was so large, it was possible to make buildings and install facilities that ordinarily could not have been made. It can easily be seen that a tipple of that kind, with the number of men it would take, would be much cheaper than one with four or five units that would give the ultimate capacity. The tonnage capacity would give a life of from 50 to 100 years at that plant.

E. W. Parker, Philadelphia, Pa.—How many seams have you worked?

Howard N. Eavenson.—There is one below water level, another about 20 ft. below the one they are now working; then about 1100 ft. above that is another, and about 600 ft. above that is another.

Graham Bright, East Pittsburgh, Pa.—I am glad to see the liberal dimensions of the car used. Those who are familiar with soft-coal conditions are accustomed to seeing cars with wheel bases of about 24 in. For the last two years the Standardization Committee of the American Mining Congress has been endeavoring to arrive at some standard dimensions for mine cars, and one dimension decided upon was that the wheel base was to be at least as great as the gage. But a height of only 22 in. seems to be a low car for so high a coal seam. What would be the objection to raising the sides a small amount and getting a little more coal into the car? The cost of underground haulage decreases rapidly with an increase in capacity.

Howard N. Eavenson.—The average height of coal in the workings is 5 ft. but there are a number of local rolls where the coal comes down to

3½ or 4 ft; that is one reason for making a small car. Besides the seam below that will range from 3½ to 4½ ft. in height, and it was desired to use the same cars there. Also we wanted to make the car as easy for the men to load as possible. As built the cars require practically no lifting of the material as the effort necessary to cast the coal to the end of the car will lift it over the side.

George S. Rice, Washington, D. C.—Is there any tendency for the coal to spill from such low-sided cars, making the roads perhaps somewhat dirtier than if the sides were a little higher? The thought in mind is the safety side because of the constant grinding of coal dust.

Howard N. Eavenson.—Such a tendency depends entirely on the method of loading; if the cars were higher, they would be loaded above the higher side just the same. If topping is required, the cars spill somewhat from the top, but it has been the practice always to keep the roads clean; that is to go over them at least once a week, or more if necessary, so that there is practically no accumulation of coal on the haulageway. Of course, that does not eliminate the dust question—there may be a little more dust than there would be under ordinary conditions, but the cars do not spill nearly as much as they do under Illinois conditions, where they are cribbed along the sides.

Mr. Ulden.—What is the cost per ton capacity of the storage for this balancing of work; and also what is the consumption of water per capita? Which forms the principal part of the wash houses—shower baths or tubs? If shower baths, how many baths are allowed for, say, 100 men?

Howard N. Eavenson.—The cost of the reinforced concrete bin at Lynch was approximately $24 per ton of capacity. This figure may be somewhat in error, as the writer has not the actual costs at hand, and some of them were not distributed as accurately as should be to determine the cost of this part of the structure.

The per capita consumption of water was estimated 50 gal. per person per day, which was about the consumption at the Pocahontas mines. Every house is supplied with running water; there is a cast-iron sink in the kitchen, and the water is used freely by all. Of course, the main bathing is done by the men in the wash houses, where showers are used exclusively. There are no tubs in the wash houses, as the welfare department is opposed to them, claiming that they are not sanitary under conditions of that kind, and in addition are much more expensive than the showers. The building contains 71 showers and 64 lavatories; that is about 4.7 showers per 100 men. It was estimated that a man would take his bath and dress in about 15 min. so that four times the seating capacity would pass through the room in an hour. That plan has worked out very well and the men do not have to wait at all. They usually begin

washing up at noon and keep arriving up to four o'clock. Locks are provided for the men, but it is preferable for the men to provide their own locks for the clothes hangers, for then they know that no other man's key will fit their lock.

George Otis Smith, Washington, D. C.—What is the accident record?

Howard N. Eavenson.—The tonnage produced per fatal accident was 124,264 tons. This figure is much lower than it should be, and it will undoubtedly be considerably increased within the next year or two. It has been the experience of the company that accident prevention is almost entirely a question of education, and as soon as the force becomes better organized and better acquainted with the company's system, the results will undoubtedly be much better.

Allan M. Rodgers, Douglas, Ariz.—Was the hot-water system successful in very cold weather?

Howard N. Eavenson.—It was only finished in the late fall of 1920 and last winter was an open one; but there is every indication that it will be successful. One other feature that I neglected to mention, was the fact that the pressure is kept on the line, by gravity, all through the summer; that is, the water is kept hot and is connected with the tank pressure without running the pumps; so that the houses always have hot water without having to maintain a fire. The temperature was about 160° but the water could be heated to 205°, so I think there is no reason why there should not be ample capacity.

Clarence T. Starr, Bethlehem, Pa.—Have you maintained the 4 tons per employee in production?

Howard N. Eavenson.—For the month of August, 1921, the last month for which data is available, the production per employee inside was 3.65 tons per day; and per total employes on the plant was 2.73 tons per day. These figures, it should be noted, are for the names on the payroll, and are not for the number of men working, which is usually from 15 to 25 per cent. less than the number on the payroll.

As the production was 148,000 tons, instead of 210,000 tons for which the plant was designed, there is little doubt that with the final organization that will be effected and at the designed capacity, the estimated figures will be reached.

Charles A. Burdick, New York City.—Have you any figures on the power used per ton produced?

Howard N. Eavenson.—The average amount of power used per ton of coal produced was 3.26 kilowatt-hours.

Ashley Planes for Handling Freight Traffic

By C. H. Stein,* New York, N. Y.

(Wilkes-Barre Meeting, September, 1921)

The Pennsylvania Legislature, on March 13, 1837, passed an act authorizing the Lehigh Coal & Navigation Co. to construct a railway to connect the North Branch Division of the Pennsylvania Canal with the slack water navigation of the Lehigh River; the navigation company accepted this act on May 10, 1837. Edwin A. Douglas, chief engineer for the navigation company, located the route for this railway from White Haven to Wilkes-Barre, which was called the Lehigh & Susquehanna Railroad. The object was to transport coal from the Wyoming Valley, of Pennsylvania, to the Atlantic seaboard. This railway was considered a marvelous piece of engineering, as it included a tunnel nearly ⅓ mi. long, and a series of inclines, or planes, up which the cars laden with coal were raised from the vicinity of Ashley to the summit of the mountain near Solomon's Gap, whence they were taken to White Haven by rail. There the coal was transferred to canal boats.

In June, 1862, floods almost completely destroyed the upper division of the canal, so on March 4, 1863, the Pennsylvania Legislature passed an act prohibiting its restoration, but granted a charter for the construction of a railway from White Haven to Mauch Chunk to connect with the line from White Haven to Wilkes-Barre, previously built. On March 16, 1864, a supplementary act was passed extending the Lehigh & Susquehanna Railroad to Easton, Pa. This completed a direct all-rail route from Wilkes-Barre to the New York harbor (by way of the Central Railroad of New Jersey).

Up to this time, all freight and passenger traffic passed over the planes, but in 1866 and 1867 there was constructed what was known as the "back track" down the mountain side between Solomon's Gap and Ashley over which all the passenger traffic and empty cars thereafter passed; it is now known as part of the main line. For the handling of the heavy coal traffic, however, this would have involved a continuous climb from Ashley, 170.27 mi., from New York, to Solomon's Gap, 157.8 mi. from New York, or a distance of 12.47 mi., and a total ascent of 1013.75 ft., composed of grades 95.61, 83, and 51.4 ft. per mi., respectively, and

* Assistant to the President, Central Railroad of New Jersey.

an average grade of 81.29 ft. per mi. for the distance of 12.47 mi. The use of the planes was, therefore, continued for all eastbound coal tonnage. On April 1, 1871, the Lehigh & Susquehanna Railroad was leased to the Central Railroad of New Jersey, which has since operated it.

The foot of the planes is near the Ashley station and the receiving yard for all the coal from the Wyoming Valley handled by the Central Railroad of New Jersey; it is also at the confluence of the main line of the railway and the Nanticoke branch, which taps one of the largest anthracite mining districts in the world. The top of the inclines is at Solomon's Gap, the summit of the main line.

The steep inclines and level stretches at the foot resemble one another, the only variation being the difference in the length and rate of grade; the steeper the grade, the shorter is the plane. The lengths and gradients are as follows:

Number of Plane	Length Feet	Grade, Per Cent.	Rise, Feet
3	5,000	5.7	269.0
2	3,000	14.65	422.2
1	3,700	9.28	334.7

This makes a total rise of 1025.9 ft. The short, so-called level stretches of track between planes 1 and 2 and 2 and 3 descend 5.4 and 6.75 ft., respectively, or a total of 12.15 ft., so as to permit the cars to drift freely to the foot of the next plane; this reduces the actual ascent of the three planes to 1013.75 ft.

The object of the planes is to overcome the small tonnage movement at low speed over heavy main-line grades for a distance of more than 12 mi. by a rapid movement of large tonnage over a distance of only 2.47 mi. over grades largely in excess of any upon which a locomotive could operate. This is accomplished by installing hoisting engines of large capacity at the head of each plane, which, by means of large cables, haul the cars to the objective point. Naturally, the plant is not now constructed as it was formerly, but the difference is only in detail.

The loaded cars are taken from the receiving yard, close to the foot of the planes, in trains of six and placed over a truck pit at the foot of plane No. 3. The footman at this point, by means of a bell, signals the engineman at the head of the plane, who applies his power and draws a truck, or "barney," from the truck pit up behind the cars. As soon as the engineman feels the barney go against the cars he speeds up his engine and the cars start their ascent of plane No. 3.

On each plane there are two parallel tracks and at the foot of each there are two parallel truck pits. The cable from one barney passes up the plane over sheaves about 22 ft. apart, then three times around drums 20 ft. in diameter, and out through the bottom of the power house

over a very large sheave, which returns the cable to a barney on the parallel track. As a result, when one barney ascends, drawing up the cars, the barney at the head of the plane descends, drops into the truck pit, passes under the six loaded cars that have been placed by the drill engine, and is ready for its ascent. Simultaneously, the first barney has reached the head of the plane, a brakeman takes charge of the cars, the engineman reverses his engine, and the first barney starts on its descent, and the second one is raised, by inclined rails, out of the pit and goes behind the cars standing to receive it. The process is now repeated, six cars moving alternately on the tracks of plane No. 3, with but slight interruption. The cars are delivered at the foot as fast as they can be drawn up the planes, or approximately so.

A lighter cable passes from the end of one barney to a high gallows from which are suspended heavy counterbalance weights, around a large bull wheel, to the other barney, for the purpose of equalizing any slackness in the main cables of either barney.

When the cars reach the top of plane No. 3, the train is cut in two, as only three cars at a time are drawn up planes Nos. 2 and 1. However, because these planes are so much shorter, the six cars are handled, in two trains, on each of these planes as rapidly as they are handled, in one train, on plane No. 3.

Formerly, all the cars were weighed, coupled together, as they passed at a speed of 2 mi. an hr., over a 200 ton track scale 117 ft. long at the head of plane No. 1. It was demonstrated that the weights taken in this manner were practically without error. A few years ago, however, it was decided that the operation would be simplified by abandoning this track scale, and dropping the cars into a receiving track, at the end of which a hump engine pulled trains of cars to a hump yard located at Penobscot, about 1 mi. east. There the cars were dropped over the scale and hump for classification purposes into a classification yard consisting of thirteen classification tracks, for the make-up of trains and further movement to the seaboard.

The cars may be drawn up the planes at speeds of from 12 to 30 mi. per hr. The plan and profiles of the planes are shown in Fig. 1.

The barney is made to pass into the truck pit, underneath the cars, and then out again behind the cars by an ingenious arrangement of the wheels of the barney and of the track devices. Each wheel has its own axle, which slides in its own sleeve independently of the others. One axle is just forward of the other and both are held in place by an equalizing bar, which adjusts itself to the gage of the wheels. The flanges, which run on the outside of the rail, are $2\frac{1}{8}$ in. deep; the treads are $6\frac{13}{16}$ in. wide. The barney weighs 7 tons; its construction is shown in Fig. 2.

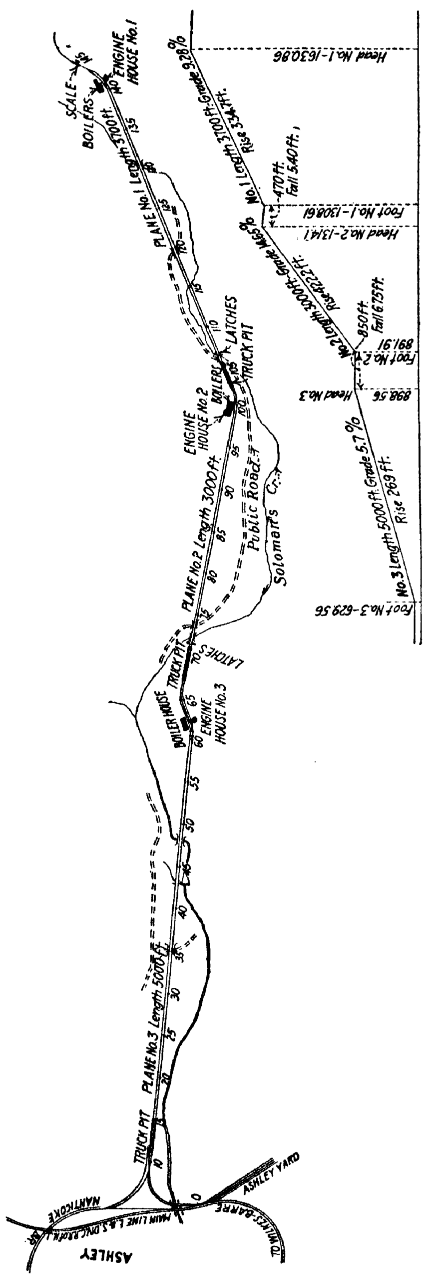

ENGINE HOUSE No.1
SCALE
BOILERS
PLANE No.1 Length 3700 ft.
ENGINE HOUSE No.2
BOILERS
LATCHES
TRUCK PIT
PLANE No.2 Length 3000 ft.
Public Road
Solomon's Cr.
TRUCK PIT
LATCHES
BOILER HOUSE
ENGINE HOUSE No.3
PLANE No.3 Length 5000 ft.
TRUCK PIT
NANTICOKE
ASHLEY
ASHLEY YARD
TO WILKES-BARRE
Head No.1-1630.86
No.1 Length 3700 ft. Grade 9.28%
Rise 334.7 ft.
470 ft.
Fall 5.40 ft.
Foot No.1-1308.61
Head No.2-1314.1
No.2 Length 3000 ft. Grade 14.65%
Rise 422.2 ft.
850 ft.
Fall 6.75 ft.
Foot No.2
891.91
Head No.3
898.56
No.3 Length 5000 ft. Grade 5.7%
Rise 269 ft.
Foot No.3-629.56

FIG. 1.

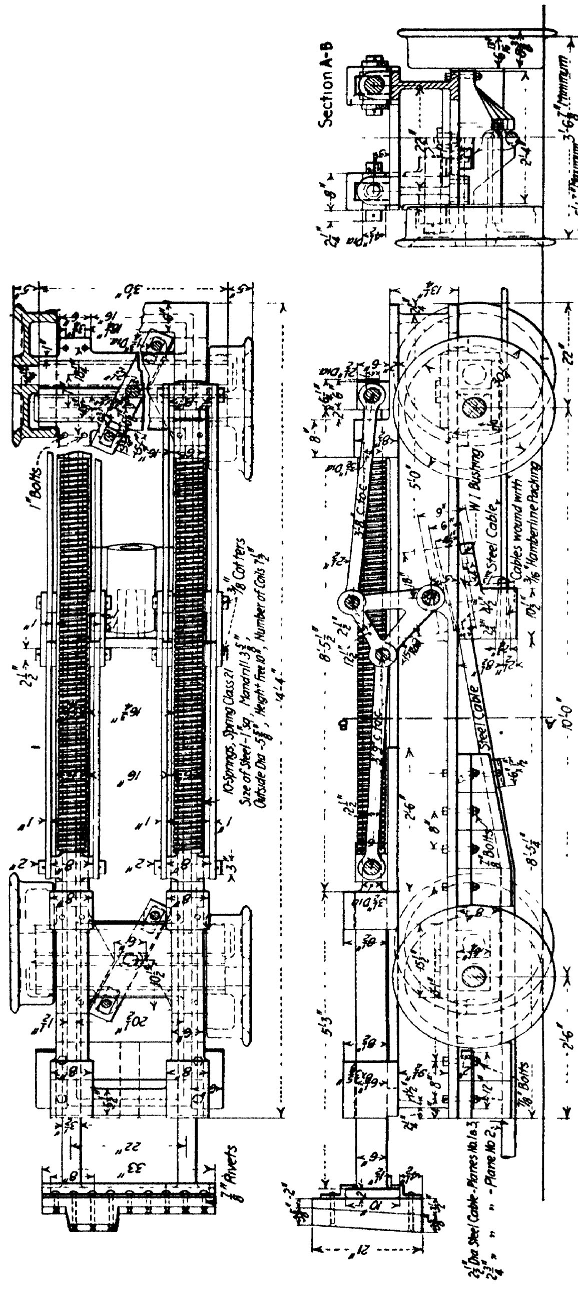

Section A-B
1" Bolts
Steel Cable
Steel Cable
W.I. Bushing
Cables wound with
Bolts

FIG. 2.

TABLE 1

	Top, or No. 1, Plane	Middle, or No. 2, Plane	Bottom, or No. 3, Plane
Length of plane, feet	3,700	3,000	5,000
Length of truck pits, feet	391	397	428
Rise per 100 ft., in feet	9.28	14.65	5.7
Total rise of each plane, feet	334.7	422.2	269
Fall between planes, feet	5 40	6.75	
Distance between planes, feet	470	850	
Total rise, Ashley to Solomon's Gap, feet			1,013.75
Total distance, Ashley to Solomon's Gap, feet			13,020
Slope angles	5° 18′	8° 20′	3° 15′
Length of main cable, feet	4,640	3,670	5,780
Diameter of main cable, inches	2½	2¾	2½
Weight per foot, pounds	10½	11⅝	10½
Length of tail rope, feet	4,023	3,320	5,520
Diameter of tail rope, inches	1½	1½	1½
Weight per foot, pounds	3¾	3¾	3¾

At first, the barney descends on the rails *A*, Fig. 3, upon which the cars travel. At *B*, though, the flanges spring out the wings and the treads ride upon rails *C*. At *D*, a deflector, which is simply a boss on a filler block, forces the flanges away from the switch point. In the pit, the flanges spring switch points *E*, so that when the movement of the barney is reversed, it ascends from the pit upon rails *F*. By these rails, the 5-ft. climb out of the pit is made in 30 ft. After the footman has given the signal, the barney ascends from the pit on rails *F*; as soon as it strikes the cars standing upon tracks *A*, the speed of the engine is increased and both barney and cars move upwards. They spring open switch *D* and the barney wheels are transferred to rails *C*. As the guard rails *G* are 3⅛ in. higher than the running rails, the flanges of the barney wheels are forced outwards, springing the wings *B* so that the barney travels upon rails *A* to the top of the plane, where there is another pit. As previously stated, as the barney on one track ascends, that on the parallel track descends. At the head of the plane the two tracks converge into a single track but they soon diverge again. In this way, the two tracks at the foot of the plane may be served with cars alternately from the tracks on the preceding plane.

At the head of each plane are a battery of boilers and a vertical engine, which furnish the power for that plane. Their horsepower rating is as follows:

	Plane No 1	Plane No 2	Plane No. 3
Horsepower of boilers	1,500	2,000	1,500
Horsepower of engines	1,200	1,200	1,200

The number of cars that can be conveniently handled over the three planes per hour averages 35; 45 cars may be handled with a little additional effort. The average tonnage for 24 hr. has been 32,424 tons.

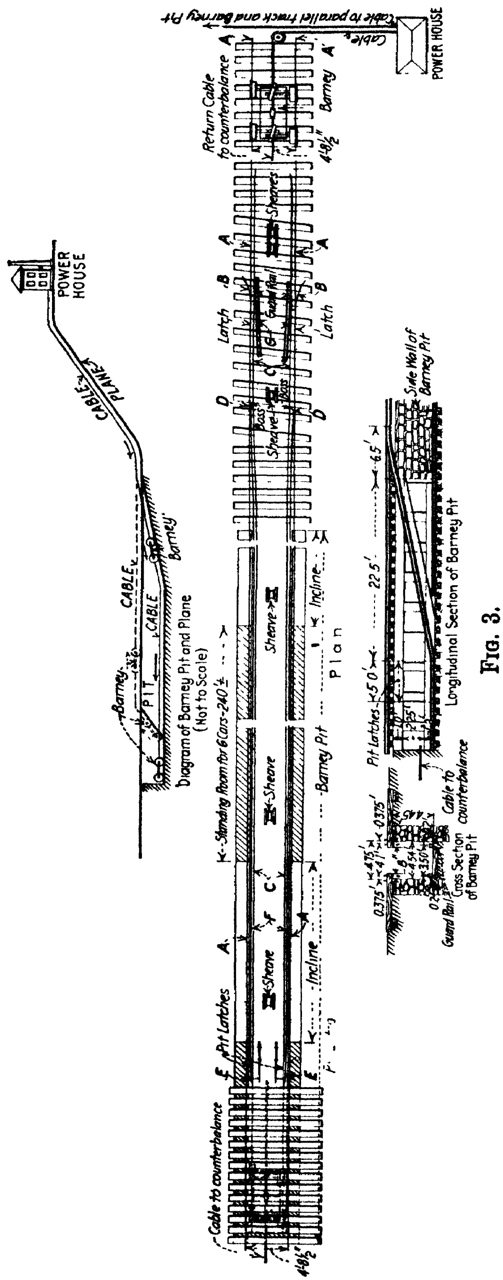

POWER HOUSE
CABLE
PLANE
Barney
CABLE
PIT
Diagram of Barney Pit and Plane
(Not to Scale)
Cable to parallel track and Barney Pit
POWER HOUSE
Return Cable to counterbalance
Barney
Sheaves
Latch
Guard Rail
Pass
Sheave
Standing Room for 6 Cars-240'±
Incline
Sheave
Barney Pit
Plan
Pit Latches
Cable to counterbalance
Side Wall of Barney Pit
Longitudinal Section of Barney Pit
Cable to counterbalance
Cross Section of Barney Pit
Guard Rail

FIG. 3.

The economy of the boiler plant has been largely increased by the installation of a patent blower system. The planes are not in constant operation throughout the year, nor the 24 hr. of a day, but the efficiency of the plant is shown by the following figures:

Number of days worked	305
Coal hauled, tons	4,827,104
Coal and weight of car, tons	6,832,935
Merchandise hauled, tons	974,800
Coal, car, and merchandise hauled in one year, tons	7,807,735

When it is considered that one of the grades is 14.65 per cent. for a distance of 3000 ft., the success of this plant suggests the application of this principle to the operation of hump yards. As many cars could be handled over the hump as could be delivered by an engine from the receiving yard. The attractive feature would be that the operations on the hump would be conducted from a point where the movement of the train would be absolutely under the control of one who could clearly observe all that was going on. It would not be necessary to pass signals to the rear end of the train, with the consequent delays and troubles. The expense of operation would not be so great as in the case of a plane, as the speed would be so much lower. There would not be the same wear and tear on machinery. The feeder engine bringing cars from the yard would have ample time to keep the truck pit supplied, hence there would be almost a continuous movement over the hump. The indications are that cars could be classified successfully by this method, and with greater economy in cost than by any method used up to this time.

DISCUSSION

Graham Bright,* East Pittsburgh, Pa.—Has the electrification of these planes been seriously considered? In many cases in the anthracite fields it has been found that the cost of operation with electricity is so much less than with steam that it pays to scrap a good engine and boiler plant and replace with electrical equipment. Possibly, twenty years ago this steam plant represented the best engineering practice and was the best system that could be installed; but here we have three boiler plants less than a mile apart, operated by three crews, and each plant with a particularly bad load factor. Now, either by one isolated electric power plant or (which probably would be much better) by purchased power, the trips on the various planes could possibly be scheduled so as to produce a good load factor and greatly cut down the cost of operation. There is little doubt but that even without such scheduling a considerable saving could be affected by electrification.

* Engineer, Westinghouse Elec. & Mfg. Co.

Mr. Stein states the capacity of No. 1 and No. 3 boiler plant as 1500 hp., No. 2 as 2000 hp. and each engine as 1200 hp. and then states that these cars can be drawn up grades anywhere from 12 to 30 mi. an hr. Either he is a little optimistic regarding the speed, or the engines are much larger than 1200 hp., because a simple calculation will show that anywhere from 4000 to 6000 hp. will be required to pull the cars up the grades at 30 mi. an hr., and from 5000 hp. to 7000 hp. will be required to accelerate the trips on the grades.

DOUGLAS BUNTING,* Wilkes-Barre, Pa.—I have not heard whether or not the railroads have considered the introduction of electric power on these plans. The speed of 30 mi. an hr. on the planes appears rather high. I am not familiar with the present boiler installations; changes have been made in the boilers since I visited the plant.

SIDNEY J. JENNINGS,† New York, N. Y.—I would like to have Mr. Stein consider the relative capital expenditure for these planes and a railroad that will answer the same purpose; the use of planes for handling railroad freight traffic is unusual. Only in two parts of the world with which I am familiar, in South Africa and in South America, were planes established to handle railroad traffic, and they were subsequently abandoned because the amount of traffic that could be handled over the planes was smaller than that which could be handled by railroad; also the expense of operating them was too great, for a larger number of skilled workmen were required to work each engine.

R. V. NORRIS,† Wikes-Barre, Pa.—Hoisting engines are not usually figured at horsepower and a statement of the horsepower of a hoisting engine does not mean much; I hope the author will add the size of these engines.

As to the comparison of the planes with the railroad, it must be remembered that the planes are very short, compared to the railroad, which, to get out of this valley, is about two times the length of the planes and necessarily has heavy grades. The planes carry practically the entire traffic of the Central Railroad of New Jersey so that there is no particular object in figuring on an increased capacity because they can carry more than the capacity that can be furnished; in other words, they are big enough.

CHARLES H. STRANGE,‡ Minersville, Pa.—We forget that the load on these planes is balanced.

R. V. NORRIS.—The load is not balanced, the barney goes down empty.

* General Superintendent, Lehigh & Wilkes-Barre Coal Co.

† Consulting Engineer.

‡ General Superintendent, Pine Hill Coal Co.

Edwin Ludlow,* New York, N. Y.—With regard to the difference of the cost of operating, some years ago when a western railroad man took charge of the Reading, he ordered the Mahanoy plane on the Reading shut down and the coal carried around by rail, for which they had an established route. Before a month had elapsed, it was found that the transportation costs had gone up and that the single track road was inadequate to handle the increased coal traffic, so the plane was again opened and has been operated ever since.

W. J. Richards,† Pottsville, Pa.—That is not the plane described here.

Edwin Ludlow.—No, but it shows the economy of using planes.

C. H. Stein (author's reply to discussion).—The question of electrifying the Ashley planes has been discussed from time to time, with the thought of substituting electric motors for the steam engines at the several power houses, to be operated from a central generating plant at Ashley. This plant would furnish both light and power to the shops, yards, buildings, crossings, etc. between Wilkes-Barre and Penobscot, including also engine terminals and outlying power units within the area of economic distribution. No definite plan, however, has been prepared from which careful estimates could be made for the consideration of economic possibilities. It is our opinion that the improvement in the generation, distribution, and conversion of electric power for units of such magnitude and diversity of purpose, has warranted the delay in the execution of the first proposals in this direction. At the same time, the constantly changing locomotive capacity or tonnage rating, and the possibility of securing a 60-ft. per mi. mountain grade for the main line, tend to reduce the relative economy of plane operation.

Twenty years, or perhaps forty years, ago it is doubtful whether any engineer would have considered, except as a last resort, the introduction of a system of inclined planes as an economic means of handling traffic of this character, with possible future increases, but as a heritage of some of the most brilliant engineers of the day this legacy, with the modifications introduced, has up to this time proved indispensable, notwithstanding the fact that the back track was constructed about 45 years ago to relieve the downward traffic.

The planes were projected at a time when some reasonable doubt was entertained as to the practicability of operating grades of 93 ft. per mile, or even considerably less, and the original plan provided for the handling of both freight and passenger business in either direction. It was not until 1886 that the change to ascending traffic only was effected.

During the periods when general repairs or renewals are necessary to the engines or vital portions of the plant, the traffic is hauled up

* Consulting Engineer.

† President, Philadelphia & Reading Coal and Iron Co.

the mountain with three or four engines to a train. If the time required for these repairs is long, the available motive power is used to its limit to prevent congestion of traffic.

Repeated compilations of comparative costs of operation have indicated that under existing conditions, both maintenance and operation of the planes have been far more economical than by way of the back track. The compilations, however, have been based on so many factors of assumption that exact costs cannot be ascertained.

As with the planes, so with the railroad, much of the present construction could be utilized in any revision plan.

The maximum loads hoisted over each plane, and the time consumed per trip, are as follows:

No. 3, or lowest plane—four 80,000-lb. and two 110,000-lb. capacity anthracite cars; average time consumed, 7 minutes.

No. 2, or middle plane—two 80,000-lb. and one 110,000-lb. capacity anthracite cars; average time consumed 3½ minutes.

No. 1, or highest plane—two 80,000-lb. and two 110,000-lb. capacity anthracite cars; average time consumed 5 minutes.

The time here referred to covers the period between trips when everything is in good working order, and no delays arise in feeding the planes at the foot or taking cars away at the top. The actual running time is, of course, considerably less between trips, Frequently the trips handled on No. 2 plane are carried through without break to No. 1 plane, thus increasing the speed of No. 1 to keep No. 2 clear. The engines on No. 2 and No. 3 planes are identical in construction and type, while the engine on No. 1 plane is more modern and of greater power.

We would not consider the load on these planes as being balanced, in the sense that the stress on the pulling cable going up the planes would be equal to the stress on the cable of barney car descending the plane at the same time.

Application of Pulverized Coal to Boilers

BY J. W. FULLER,* ALLENTOWN, PA.

(Wilkes-Barre Meeting, September, 1921)

DURING the last 20 years, experimenters have sought to utilize pulverized coal in boiler plants, but refractory and slag troubles have usually overbalanced any gains in efficiency that were obtained. These troubles were caused largely by lack of knowledge of the proper methods of coal preparation and lack of a furnace design that would be adaptable to pulverized-coal firing. When this information was obtained the results were so much more satisfactory that within the last 5 years approximately 80,000 boiler hp. have been installed in plants ranging from 1000 to 20,000 boiler hp. rated capacity, which are operating at an average of 174 per cent. of rating.

The first commercial installation for burning pulverized coal under boilers was made at the Parsons shops of the M. K. & T. railroad in 1916. Of eight 250-hp. water-tube boilers thus fitted, six were operated continuously for 3 yr., when oil was obtained at such a low price as to warrant its use. Since then, this plant has been operating alternately on pulverized coal and oil, depending on prevailing prices. Recently gas has been discovered within 20 mi. of this plant, so that this fuel may also be used. No change in furnace design was made when changing from pulverized coal to oil or gas firing.

One of the principal advantages of this method of firing is its ability to utilize, with maximum efficiency, all kinds of coal, irrespective of quality or physical characteristics. In fact, quite a number of the present pulverized-coal installations were made for the purpose of utilizing coal that could not be marketed or used in any other way. The Pacific Coast Coal Co., at Black Diamond, Wash., formerly hand fired the boilers, using approximately 125 tons of prepared steam-size coal daily. The Black Diamond steam coal is the best quality of coal for steaming purposes mined in that section of the country, and there is a market for all that can be produced. By installing a pulverizing plant, it was possible to substitute coal of a size and quality that had no commercial value, so that 125 tons daily of the best steam coal formerly used at the mines is now marketed.

This company made a similar installation, at its New Castle, Wash., mine with approximately similar results. Each of these two plants has a

*President, Fuller Engineering Co.

pulverizing capacity of 4 tons per hour and is so designed and arranged that but one man is required for operation. The labor cost is reduced about 50 per cent. and considering all factors both plants paid for themselves in six months time.

The Puget Sound Power & Light Co., in 1918, replaced fuel oil with pulverized washery sludge in its central steam-heating plant in Seattle. This company owns property on which several hundred thousand tons of waste coal had accumulated. Every effort was made to utilize this fuel, but without success until pulverization was tried. The plant furnishes steam heat to the principal buildings in the business section of the city and has operated without any interruption in steam service during the past three years, using a very poor grade of coal.

The quality of coal used by the Puget Sound Power & Light Co., like that used at the mines of the Pacific Coast Coal Co., is a Washington lignite washery sludge containing 20 to 24 per cent. moisture, as received at the preparation plant, and in size is similar to anthracite culm. In both of these plants, the sludge is dried to between 3 and 5 per cent. moisture, after which it is pulverized in Fuller screen-type mills, and then delivered to the boilers. The average approximate analysis of this coal, as fired, is as follows:

Water, per cent	3.81
Volatile matter, per cent	39.81
Fixed carbon, per cent	41.40
Ash, per cent	14.98

At the Allentown Portland Cement Co.'s boiler plant., Evansville, Pa., pulverized anthracite culm is being burned continuously under one 400-hp. Rust-type boiler. This culm carries from 20 to 30 per cent. ash and from 4 to 8 per cent. volatile matter. This plant has also burned successfully various low-volatile fuels, such as Rhode Island graphite coal, carbocoal, the poorer grades of coal breeze, and anthracite waste containing 65 per cent. of ash.

During 1919, a preparation plant and pulverized-coal equipment for firing thirteen boilers was installed at the British Columbia Sugar Refining Co.'s plant, Vancouver, B. C. In this plant, California fuel oil was replaced with pulverized Nanaimo slack coal from Vancouver Island. Nanaimo coal is particularly high in ash, averaging 28 to 30 per cent. A typical sample of dried coal, as fired, contains:

Moisture, per cent	1.1
Volatile matter, per cent	32.8
Fixed carbon, per cent	37.7
Ash, per cent	28.4
Calorific value, B.t.u	9364

The coal is pulverized in Fuller screen-type mills until 81.1 per cent. passes through 200 mesh and 92.25 per cent. passes through a 100-mesh sieve. The acceptance test was run on the 504-hp. Badenhausen boilers, developing 83.3 per cent. combined furnace and boiler efficiency. This plant has been in continuous operation since it was installed and the management has recently added two 500-hp. Stirling-type boilers equipped for burning pulverized coal.

About a year ago the Southern Anthracite Coal Mining Co. at Russellville, Ark., equipped one 250-hp. Babcock & Wilcox boiler to operate with anthracite refuse at 200 per cent. of rating. Its operation has been so successful that negotiations are now under way toward making additional installations in the district, which will use a grade of anthracite that heretofore had no commercial value.

Recently there has been put into service the largest pulverized-coal-fired power plant in the world, namely the Lakeside plant of the Milwaukee Electric Railway, Light & Power Co. The first 20,000 kw. unit of an ultimate 200,000 kw. plant is operating exclusively on pulverized Illinois coal having the following approximate analysis, as fired:

Moisture, per cent	2 42
Volatile matter, per cent	37.53
Fixed carbon, per cent	50 87
Ash, per cent	11 6
Calorific value, B.t.u.	11,500

This installation was made after exhaustive tests and investigations at the Oneida Street plant to determine the most efficient and economical method of firing. The installation comprises eight 1300-hp. Edgemoor boilers. On 48-hr. tests, these boilers showed an overall efficiency of 88.4 per cent. at 203 per cent. of rating.

The W. B. Uihlein Co., manufacturers of food products in Milwaukee, put in four new 500-hp. Wickes boilers, which were supposed to be stoker fired, but after investigating the economies of pulverized coal this company disposed of the stokers and substituted pulverized-coal equipment.

The Susquehanna Collieries Co. has burned pulverized anthracite culm very successfully. Its first installation was made in 1918, under a 250-hp. Babcock & Wilcox boiler. From the results of this operation, the company built a new power house at Lytle, Pa., which consists of six 300-hp. Babcock & Wilcox boilers with furnaces arranged for 200 per cent. rating. This plant has been operating about two years and the company recently put into operation another new plant at Lykens, Pa., consisting of six 500-hp. and six 600-hp. boilers designed to operate at 200 per cent. rating. The 500-hp. boilers furnish high-pressure steam to the turbine and the 600-hp. boilers furnish low-pressure steam for mine operation. The steam lines are so arranged, however, that any of the

low-pressure boilers can be used for furnishing high-pressure steam, or vice versa. This company has available several million tons of anthracite culm. The power station and drying and pulverizing plant are located on the side of a hill above the top of the culm bank. The culm is pumped to a dewatering plant located above the coal plant. Space is allowed for considerable storage at this point and the culm is then conveyed to the drying and pulverizing plant. The ultimate capacity will be 400 to 500 tons per day.

In installations contemplating the use of high-ash coal or coal containing a large amount of slate and non-combustible, it is frequently possible in some way to reduce the amount of non-combustible material at a cost that is more than offset by the saving due to the decreased burden on the pulverizing and coal-handling machinery, together with the decreased cost involved in disposing of the ash. However, where ash can be sluiced from the boiler furnace, as is often possible at mines having large quantities of water available, the high-ash coals can be burned with little cost of ash handling.

The foregoing instances are only a few of the many successful installations burning pulverized coal that were made under various types of boilers and using many fuels. The cost of good coal will probably steadily increase and it is an economic duty to conserve as far as possible the better grades by burning them in a pulverized form, as well as to replace them when possible with fuels having a low market value, or which even entail an expense to handle and are a liability to the producers.

DISCUSSION

George S. Rice,* Washington, D. C.—How does the cost of pulverizing anthracite compare with that of pulverizing bituminous coal suitable for this process?

H. A. Reichenbach,† Allentown, Pa.—The cost will run several cents higher per ton, due to the greater hardness of anthracite. The capacity of the pulverizing mills is somewhat lowered in crushing anthracite, which will increase the power cost slightly. The actual cost of pulverizing and preparing the coal, however, will depend on local conditions altogether; the size of the plant, for instance, and the value placed on the coal used.

E. M. Chance,‡ Philadelphia, Pa.—What power will it take to reduce a ton of No. 3 buckwheat to the required fineness?

H. A. Reichenbach.—I have not seen any figures on that particular grade of coal, but I would say roughly it would require about 19 or 20 kw.-hr. per ton for the crushing, to the complete delivery to the boilers.

* Chief Mining Engineer, U. S. Bureau of Mines.

† Fuller Engineering Co.

‡ Engineering Manager, Day & Zimmermann.

H. M. CHANCE,* Philadelphia, Pa.—What relation does that 19 to 20 kw.-hr. bear to the energy required to reduce ordinary bituminous coal to the finest grade for dust firing?

H. A. REICHENBACH.—Bituminous coal will require from 16 to 18 kw.-hr. per ton.

H. M. CHANCE.—I have had no experience in the pulverizing of anthracite but have some data that may have a bearing on the subject. Some years ago the Scranton Mine Caving Commission inaugurated a series of experiments to determine the strength of anthracite, in order to learn the strength of pillars of coal left to support the roof. Several hundred very carefully prepared samples were obtained and experimentally crushed in different laboratories and the results of those determinations were published by the Commission. If I recall rightly, the average crushing strength of anthracite was 2000 to 3000 lb. per sq. in.; though some samples had a crushing strength of 5000 to 6000 lb. per sq. in. Later it occurred to me that the crushing strength of anthracite might depend on the size of the piece, because the larger the piece the more fracture planes it would be likely to contain. To determine this, we first prepared samples of ½-in. cubes and found these had a crushing strength of 7000 to 8000 lb. per sq. in. We then prepared ¼-in. cubes and finally made ⅛-in. cubes, which had a crushing strength of 19,000 to 20,000 lb. per sq. in. In other words, the inherent strength of anthracite, apparently, is thus found to be nearly equal to that of granite. If that is true, in some cases it might cost as much to reduce anthracite to powder as it does to reduce granite to the same fineness.

H. A. REICHENBACH.—I have no acquaintance with the pulverization of granite, but the actual crushing costs of anthracite are available in several plants where this crushing is done commercially.

GEORGE H. ASHLEY, Harrisburg, Pa.—Are the costs given those of crushing light-weight coal or hard anthracite? The Lykens coal is very tender and while it passes commercially as anthracite, it is a quite different product.

H. A. REICHENBACH.—I have no figures on the Lykens coal; there may be some available.

ROBERT A. QUIN,† Wilkes-Barre, Pa.—We are now grinding hard ash coal as well as the Lykens Valley red ash coal and burning it, too.

CHAS. W. LOTZ, Pottsville, Pa.—We have successfully pulverized Lykens coal with as little as 20 kw.-hr. per ton; but the average run would be nearly 40 or 50 kw.-hr. throughout the whole plant. The coals on

* Mining Engineer.

† General Manager, Susquehanna Collieries Co.

which we carried out our experiments were in the Lykens and the White Ash district farther east in the Schuylkill field. We have made some experiments on the coal of the Shamokin district, but they are merely laboratory experiments; and we have no practical results with this coal.

On the general subject of pulverizing and burning anthracite, I feel safe in saying that so far as the burning is concerned, we have commercially solved the problem. If you consider only the softer Lykens coal, pulverizing the coal is easy. The harder coals of the middle part of the Schuylkill district have been successfully pulverized, but it is not such an easy proposition as the Lykens material.

THOS. E. FISHER, Hackensack, N. J.—Does the 30 to 40 hp. considered necessary to grind this coal, include power for the drier, the fan, and the screw-feed, plus the pulverizing?

CHARLES W. LOTZ.—Yes.

GEORGE S. RICE.—Is there any material difference in the length of flame between the Lykens Valley coal and the harder coal from the other districts?

CHAS. W. LOTZ.—Only in this way; ignition is a little more difficult to establish in the coals from the other districts, which contain a little less volatile matter. The difference in the length of flame, however, does not exceed 15 in. We use a comparatively short flame and attempt to (and do) control its length so as to get as close to an underfeed stoker condition as possible; that is, very quick combustion.

GEORGE S. RICE.—In that case, why is the firebox so much larger than that used in the bituminous installations?

CHARLES W. LOTZ.—As a matter of fact, the proportions are almost exactly the same, if you understand by proportions the number of cubic feet of furnace volume per pound of coal burned per hour; that is, by checking certain bituminous installations against our own, we have found that our furnace volume is just about the same as the bituminous.

GEORGE S. RICE.—My impression has been that it was considered desirable in the use of a long-flame fuel, like lignite or bituminous coal, to have a fairly large firebox, so as to prevent the flame being cooled or extinguished by contact with the relatively cool boiler surfaces, but I would have supposed that with the shorter flame of anthracite there would be less loss of available heat in bringing the flame closer to the boiler surface; in other words, have a smaller firebox. What has been the observation in these experiments regarding this point?

H. A. REICHENBACH.—Anthracite is fired vertically, and there are two flows of gas through the furnace, so that anthracite firing, compared to bituminous firing, requires really 25 per cent. more furnace volume

because there is approximately 25 per cent. more volume of gas per developed horsepower in the furnace.

CHAS. W. LOTZ.—I would like to say in answer to Mr. Chance, that out of a long series of tests made on one pulverizer, so that the machine could not influence the results, the pulverizing resistance of the coal actually decreases with the size, up to a certain limit, of course.

H. M. CHANCE.—These experiments are, perhaps, of more interest to geologists than to the coal operators. At the same time, it does not follow necessarily, that because those tests show that the inherent strength of anthracite is nearly that of granite, that it would take the same power to crush it, for anthracite may be more brittle than granite; in which case, the energy necessary to pulverize it would not be so great. I have stated these facts as to the relative strength of anthracite, when reduced to small particles, as an explanation, perhaps, of this difference in the horsepower required to pulverize anthracite as compared with that required to pulverize bituminous coal.

I might say that the samples on which these tests were made were purposely selected from samples of hard, firm coal—coal that was brittle with almost a glassy fracture and very much harder than the average anthracite. The same kind of coal was used that is sometimes used in making desk ornaments from anthracite.

CHAS. W. LOTZ.—It might be of interest to know some of the things that usually are listed in engineers' handbooks as "Don'ts."

The first would be: Don't buy or attempt to use low-grade refractories. The combustion of this pulverized material is accomplished only at a temperature so high that all of the ash in the coal is, at some point in the flame, in a molten condition. The Bureau of Mines reports the melting point of ash is about 3000° F. A large part of the heat generated by the combustion is dissipated by radiation, some of which gets into the refractories. So if the refractories are incapable of withstanding that high temperature, in the presence of the ash, you are almost certain to run into difficulties.

Leaving the immediate boiler, another "don't" would be: Don't consider the fact that because a plant will pulverize one class of coal that it is capable of pulverizing every kind of anthracite; in other words, it seems to be necessary to consider each mine or possibly each locality as a separate problem.

Another would be: So far as we have been able to determine, don't consider that you can get the same results, so far as dryness in your dry coal is concerned, as are now being accomplished by the soft-coal people. That statement, however, must be considered as tentative.

We may find that we can successfully handle pulverized anthracite carrying a great deal more moisture than the present upper limit.

Still another would be: Don't think that low-grade coals immediately become available through this pulverizing process. It has been our experience that though the low-grade coals may be made available, they are not available as they exist, for the reason that the pulverizing difficulties are greatly increased by the ash in the coal, or the material which is reported as ash, and that there is a great deal more refuse to handle from the furnace, with its consequent destructive action to the refractories, and that the efficiency of the installation as a whole drops off with an increase in ash content in the coal.

John Stevenson, Jr., Sharon, Pa.—Is the pulverized coal good for anything else than being consumed under boilers? In a steel works with which I have some connection, an ingenious employe dropped the pulverized coal into the almost invisible gas passing from the byproduct coke ovens to the open-hearth and, to the surprise of the melter, it made the gas more visible for the melter and increased the product of the furnace,

H. A. Reichenbach.—There are a number of installations, other than boilers, where pulverized coal is being used successfully. I did not mean to confine myself entirely to the boiler problem. It is being used in several open-hearth furnaces, in forge heating furnaces, copper furnaces, heating and melting furnaces, and of course we all know that practically 90 per cent. of the cement burned in the country is being burned with pulverized coal.

Thomas F. Downing, Jr., Lundale, W. Va.—I understand, from the discussion, that the successful burning of this fuel depends largely on the quality of the coal used. It has been stated that two kinds of coal are burned in the same furnace, what is the volatility of the bituminous coal burned, and what is the relative value of the bituminous and the anthracite dust?

A. E. Douglass, Allentown, Pa.—At the Allentown plant, we use about 150 tons of coal a day; bituminous coal is used for the rotary kilns, the anthracite is used only under the boilers. At times, as when the supply of anthracite is exhausted, bituminous coal is burned under this boiler. Both kinds of coal are not used at one time. The two systems of burning were installed for an emergency and to take advantage of the cheaper fuel regardless of whether it is high or low in volatile matter.

Thomas F. Downing, Jr.—The point I was trying to bring out was this: What is the difference in the quantity of coal consumed in each case?

A. E. Douglass.—That depends on the calorific value of the coal. The same rating can be obtained with either bituminous or anthracite.

Of course, on some grades of coke breeze or anthracite, it may take 3½ lb. of coal per boiler horsepower, while on bituminous coal it may require 3 lb. This depends on the B. t. u. value of the fuel. As to efficiency, on actual tests on the boilers, the efficiencies have been practically the same, within a fraction of a per cent. (as near as you can get any two tests to check), on bituminous as on anthracite. In other words, the best results on that boiler were 78 per cent. on bituminous and 77.8 per cent. on anthracite, that is, boiler and furnace efficiency. I refer to the old type two-pass boiler. Bituminous coal ran 8 per cent. ash in that highest test and the anthracite ran 14.6 per cent.

That plant has three pulverizers and two driers; one pulverizer and one drier are used for the boilers and the others for the kilns. For a period of about 8 mo., which is all we can take on anthracite, the total cost of preparing bituminous coal, assuming that the labor was equally divided, or prorated, according to the tonnage put through the plant, has been 60 cents and for anthracite 72 cents. The power consumption was 14 kw.-hr. per ton for the bituminous coal of the pulverizers, with 17½ kw. for the entire plant; while with anthracite, it has been 18 kw.-hr. on the pulverizer, and 21½ kw. on the entire plant; that is about 25 per cent. increase in power.

George Otis Smith,* Washington, D. C.—Without discounting for a moment the wonderful success, from the combustion point of view, of this utilization of pulverized coal, the questions asked here and their answers have brought out the fact that considerable power (and that means considerable coal) must be used in pulverizing this fuel. We have had it brought to our attention several times that the cost of mining anthracite, expressed in terms of coal, is 8, 10 or even 12 per cent. Now, as I understand it, we have a similar high cost, and possibly 50 per cent. higher cost, in preparing this pulverized coal; in other words, it takes as much, or possibly 50 per cent. more, coal to prepare this coal after it has been mined, as it did to perform all of the operations connected with mining. That, of course, may lead to several conclusions: one is that the coal miners are not so terribly wasteful in their use of mechanical power, if they can take the coal, so far as mechanical power is concerned, at the face in the mines and mine and prepare it for market with an expenditure of 10 per cent.; or if, as Mr. Thomas has stated, under the best conditions, only 5 per cent., whereas 10 or more per cent. of the coal is necesssary to prepare it for this special utilization. Is the gain in combustion worth that cost?

Robert A. Quin.—We started to pulverize anthracite coal and silt about two years ago, but up to the present time on account of still being

* Director, U. S. Geological Survey

in the experimental stage we cannot give any definite figures as to the cost of pulverization on a tonnage basis. Commenting on the commercial side of the method of utilization of this fuel, we are not prepared to give the Institute anything of interest. The statements of Mr. Lotz as to the various problems we encountered and troubles we have met, warrant me in stating that we are fast reaching the point where we are not only going to accomplish our aims on a practical basis, but also on a commercial basis.

Time Factor in Depletion of Mines

By John W. Roberts, C. P. A., N. C., Clarksburg, W. Va.

(New York Meeting, February, 1921)

The Federal income tax law permits as a deduction in determining net income "in the case of mines, . . . a reasonable allowance for depletion and for depreciation of improvements, according to the peculiar conditions in each case, based upon cost, including cost of development not otherwise deducted: Provided, that in the case of such properties acquired prior to Mar. 1, 1913, the fair market value of the property (or the taxpayer's interest therein) on that date shall be taken in lieu of cost up to that date."

The law provides for a reasonable allowance but the regulations provide for a definite method of computation that possesses more the virtue of simplicity than that of reasonableness. Article 210 of Regulations 45 reads: "When the cost or value as of Mar. 1, 1913, or within 30 days after the date of discovery of the property shall have been determined, and the number of mineral units in the property as of the date of acquisition or valuation shall have been estimated, the division of the former amount by the latter figure will give the unit value for purposes of depletion, and the depletion allowance for the taxable year may be computed by multiplying such unit value by the number of units of mineral extracted during the year."

This is simple, but is it reasonable? The intent of the law is clearly that the cost of the ore in the ground shall be considered as part of the cost of the same ore when removed and sold. The regulations fail to carry out that intent, because of the unwarranted assumption that all the ore in the mine cost the same amount per ton. This is what gives the Treasury Department's method its simplicity, but at the same time robs it of its reasonableness because the assumption is contrary to fact.

It is self-evident that a ton of ore near the surface or exposed by present workings is worth much more than a ton buried so far down in the earth that it cannot be removed for many years. If no improvements in mining methods were expected, the value of deeply buried ore would be further reduced by the greater expense that would be required to

bring it to the surface. Experience seems to justify the belief, however, that the improvements in methods are likely to keep pace with the difficulties encountered in the majority of cases.

But whether or not inventions may be expected to offset, in some measure, the increasing difficulties of greater depth, there is nothing to compensate for the time element, which is always an important factor. No extensive deposit of coal or ore can be mined in a day or a year. Almost invariably before a deal in mining properties is consummated the purchaser has, through specialists, made exhaustive studies of the extent and quality of the deposit and the most economical methods of its exploitation. It would not be economical to mine one ton a year and it would ordinarily be impossible to mine the whole deposit in a year. But the plan adopted is usually based on the annual production of a certain definite quantity. Ordinarily the purchaser not only has these plans, but has concrete evidence that would convince any fairminded and disinterested person that in purchasing the property the price he was willing to pay was based upon these engineering reports.

Suppose that the mine is known to contain 1,000,000 tons of ore, and that the plans call for the mining of 20,000 tons per annum, indicating a life of 50 years for the mine, and that the mineral deposits are purchased for $100,000. Although the average cost per ton is ten cents, that figure does not apply to the ore near the mouth nor to that deeply buried.

To determine the cost of the several tons, indicate by V the value of a ton exposed and minable today, and assume an interest rate of 6 per cent., and, to avoid the unnecessary intricacies of computation, also assume that each year's production comes at the end of the year. The cost of the ore to be mined the first year will then be $20{,}000\ V \times \frac{1}{1.06}$. The cost of the ore to be mined the second year will be $20.000\ V \times \frac{1}{(1.06)^2}$. The cost of the ore to be mined the third year will be $20{,}000\ V \times \frac{1}{(1.06)^3}$ etc. The sum of this series for fifty terms constitutes the cost of the entire deposit. Therefore

$$\$100{,}000 = 20{,}000\ V \times \left[\frac{1}{1.06} + \frac{1}{(1.06)^2} + \frac{1}{(1.06)^3} + \cdots + \frac{1}{(1.06)^{50}}\right]$$

The series in the bracket is equivalent to the present value of an annuity of $1 for 50 years at 6 per cent., which is readily computed as $15.761. The equation is therefore reduced to $100,000 = 20,000 V × $15.761. Solving for V, the cost of one ton minable today is found to be $0.3172; and from this the cost of the 20,000 tons to be mined each year may be computed as follows:

	Cost Per Ton	Cost of 20,000 Tons
1st 20,000 tons	$\$0.3172 \div (1.06) = \$0.2992$	$5,984
2d 20,000 tons	$0.3172 \div (1.06)^2 = 0.2823$	5,646
3d 20,000 tons	$0.3172 \div (1.06)^3 = 0.2663$	5,326
4th 20,000 tons	$0.3172 \div (1.06)^4 = 0.2512$	5,024
5th 20,000 tons	$0.3172 \div (1.06)^5 = 0.2370$	4,740
10th 20,000 tons	$0.3172 \div (1.06)^{10} = 0.1772$	3,544
15th 20,000 tons	$0.3172 \div (1.06)^{15} = 0.1324$	2,648
20th 20,000 tons	$0.3172 \div (1.06)^{20} = 0.09897$	1,979
30th 20,000 tons	$0.3172 \div (1.06)^{30} = 0.05528$	1,106
40th 20,000 tons	$0.3172 \div (1.06)^{40} = 0.03088$	618
50th 20,000 tons	$0.3172 \div (1.06)^{50} = 0.01725$	345

If this table were filled in for the full 50 years the last column would total $100,000, which is the cost of the entire deposit.

If an interest rate of 8 per cent. were adopted the cost of a ton minable today would be $0.4087, and the cost of the respective groups of 20,000 tons would range from $0.3784 to $0.00871 per ton.

Other properties than mines are acquired for lump sums, and in those cases it is conceded to be the right and duty of the purchaser to analyze his cost and set up in separate accounts a fair apportionment of it. Thus, a taxpayer may buy for a lump sum a going store with all its assets and liabilities. He is expected to apportion this cost and set up separately, in his books, the land, buildings, furniture, merchandise, accounts receivable, good will, accounts payable, etc. The apportionment must be fair and the net total must agree with the aggregate cost.

As another example, a merchant buys for a lump sum a miscellaneous lot of hides which he sorts into numerous grades. The best hides may be worth many times as much as the poorest, but is he required to cost them all at the average cost? No. He is expected to apportion the cost on the basis of quality and value. The apportionment must be fair and the total of the costs thus allocated to the several grades must agree with the aggregate cost. If the hide merchant sold all the poorer grades, but had the best hides on inventory at the end of the year and priced these at the average cost, there is little doubt but that the Treasury Department would compel him to use a higher figure and would assess an additional tax.

With the mining company, the shoe is on the other foot. In the nature of its operations it mines and sells first the most accessible ore—the ore that really cost most—and is then asked to carry the less accessible ore on its balance sheet at the average cost. The law permits a reasonable allowance for depletion, and any mining company that makes a reasonable and fair apportionment of the cost of its mineral deposits should be accorded the same fair treatment that is accorded to the merchant of hides.

The Treasury Department must recognize that its regulations are valid only in so far as they may be consistent with the law. In framing the general regulations, it was necessary for the Commissioner to provide a rule applicable even in cases where no evidence was submitted as to the probable rate of removal of the minerals. The law, however, requires that the depletion allowance be made "according to the peculiar conditions in each case" and the Treasury Department can hardly contend that it is complying with the law if it disregards the peculiar conditions of a case where such conditions are properly shown.

In the recent stock-dividend decision, as well as in prior decisions, the Supreme Court has refused to uphold the levy of an income tax where the existence of the income is debatable, and there would be excellent ground for expecting the courts to recognize the factor of time as bearing upon the apportionment of depletion charge even if the Internal Revenue Bureau should adhere to the letter of its regulations.

Capitalization of Mine Development

BY JOHN B. DILWORTH, E. M., PHILADELPHIA, PA.

(Wilkes-Barre Meeting, September, 1921)

THE word "development," as used in connection with mining, is a rather general term and in most instances must be qualified or explained before the exact thought in the mind of the user is made clear. For instance, by the development of a mining district is often meant its entire facilities of mineral production, such as railroads, towns, mine plants, etc. In speaking of the development of a property its prospecting by test pits or diamond drill borings may be referred to; or it may be the mines, strip pits or quarries from which the coal, ore or rock is won. But, as a rule, when the development of an individual mine is spoken of the idea in mind is the system of underground workings by means of which the mineral is recovered and transported to the surface. It is this character of development that is considered here.

The discussion will be confined to the ordinary room-and-pillar system of mining practically horizontal coal seams that is used in most bituminous coal mines in this country, and to only two phases of this subject; *viz.*, what is properly included under the head of "Development" as it appears on the books of a coal company or the inventories of an examining engineer, and how its "value" should be determined by the accountant or engineer.

No absolute rule or definite agreement covers these points, yet these questions are far more than academic. Development is often an important part of a mining company's assets, affecting the amount of its invested capital (and hence its rate of taxation under present Federal income tax laws) as well as the value of its total assets and the amounts chargeable to cost of coal and to depreciation.

MINE DEVELOPMENT ACCOUNT

Perhaps the majority of mine accountants will agree that the mine development account is that capital account which shows the extraordinary expenditures incurred in opening and expanding the underground workings of a mine until it has reached its normal production, the expenditures being those made for shafts, slopes, rock tunnels, drift mouths and such relatively permanent work; also payments over and above ordinary mining costs made for driving entries, air courses and break-throughs; for

timbering, grading haulageways, etc. Yet this definition does not express the universal practice and is subject to various interpretations.

Closing Mine Development Account

One common departure from this rule is with regard to the period of the mine's operation when the development account should be closed. Some think this account should be closed not when the mine has reached its normal production but when the returns from the coal produced equal the cost of production. Others prefer to limit the development period to the time when the output has reached a certain percentage (say 25 or 50 per cent.) of the anticipated normal production. But the best practice would seem to be to continue the development account until the mine is prepared to afford a production that it may reasonably be expected to maintain over a period of years.

This is evidently the view held by the committee appointed by the National Coal Association to consider a "Standard System of Accounting and Analysis of Cost Production," for its report states "After a coal mine has been developed and equipped to its planned output capacity, charges to its capital account should cease," etc. This report also quotes as follows from United States Treasury Regulation No. 45, Art. 222:

> In the case of mining operations all expenditures for plant, equipment, development, rent and royalty prior to production, and thereafter all major items of plant and equipment, shall be charged to capital account for purposes of depletion and depreciation. After a mine has been developed and equipped to its normal and regular output capacity, however, the cost of additional minor items of equipment and plant * * * may be charged to current expenses of operation.

While the limits of the development period are not altogether clear, the last sentence seems to indicate that the writer considers a mine developed when it has reached its normal and regular output capacity.

Items that Form Extraordinary Expenditures

It is generally agreed that the total actual cost of such work as the sinking of shafts, driving of rock slopes or tunnels and construction of drift portals should be charged to the mine development account, as it is of a more or less permanent nature comparable to plant construction and installation. These will be referred to hereafter as class A expenditures.

But when preliminary work intimately connected with the recovery of coal is involved and where the outlay is partly or wholly offset by income from the sale of this coal, opinions differ as to how much of the expense is properly chargeable to development. Examples of such expenses are, compensation over and above the regular room rate paid to miners for

coal from narrow places such as entries, air courses, crosscuts, and room necks; yardage paid for removing rock, bone, or other non-commercial material from roof or floor of the seam in such places in the regular course of mining to provide sufficient head room; extra payments made for driving entries through clay veins or local rolls, etc. These will be called class B expenditures.

Expenses of an intermediate nature between classes A and B are those of grading haulage entries by cutting through rolls in roof and floor; driving entries through zones of thin, impure, or faulty coal; timbering or otherwise securing a safe roof in entries; constructing brattices, over castsand air doors.

Some operators maintain that the development account should be charged with the entire cost of all such work during the development period and credited with receipts from the sale of coal produced, the difference being the cost and value of development. While this method, in a sense, shows the net cost of the work it will rarely give its true value, for the selling price of the coal varies with market conditions and bears no direct relation to the value of development. Thus, the development account of a mine opened during a period of high coal prices would show a much lower net charge than this account for a similar mine opened when coal was cheap.

Another method charges to development all payments above the room rate made for narrow work, yardage and other operations in class B. It assumes that all mining expenses over and above the cost of room coal during the development period are for additions or improvements to the mine and therefore chargeable to this division of the capital account. This method, while preferable to the preceding one, allows too great a profit on the tonnage sold by placing its cost at that of the cheapest coal that the mine affords.

The best method seems to be to charge to the development account all work in classes A and B during the period of development and credit it with the normal cost of the coal recovered. The debit balance, when the development period is passed, will be the net cost or value of mine development. By *normal cost* is meant the product of the number of tons recovered multiplied by the average cost of producing a ton of coal when the mine is in full operation, comprising expenses of mining, dead work, hauling, tipple, ventilation, drainage, power, superintendence, and such items as constitute regular expenses of mining. It may be claimed that this practice involves the use of an estimated figure, normal cost per ton, in the case of a new mine. But this is not a serious objection for, in the case of most coal mines, an experienced engineer or operator, given the wage scale, can estimate quite closely what this cost will be.

The advantages of this method are its independence of the market price of coal or the cost of room coal and that it gives a figure that really represents the extra cost of preparing the workings to produce their nor-

mal tonnage. Should the price of coal be high or low during the development period the operation would show the same profit or loss it would have made on an equal tonnage had it passed the development stage; but the development account would be unaffected.

Classifying and Valuing Mine Development

Some appraisers give a fixed value per foot to the main entries alone or to special parts of them; others classify the entries according to their condition and use and assign a separate per-foot value to each class; still others calculate that it costs a certain amount to develop a working face and give the product of this figure by the number of active faces as the value of the mine's development. Any of these methods will give reasonably correct results if their basic factors are carefully derived and proper allowances made for depreciation through exhaustion of tributary coal. But they are not easily adaptible to the variable conditions found in different mines, they leave much to the individual judgment of the appraiser and can be checked only with difficulty by an independent investigator. A better plan is to use the method, already indicated, of estimating the excess cost of underground work over normal mining costs for the period of development; *i.e.*, until the workings have sufficient active faces to afford the mine's normal output.

Illustration of a Development Account at a New Mine

Assume an area of coal 4 ft. thick on which it is proposed to place a plant of sufficient capacity to produce a normal output of 1000 tons a day, or 230,000 tons annually. Of course, the maximum capacity would probably be 50 per cent. greater because no mine runs uniformly at its highest efficiency, but the production figure to be decided upon is that which the entire operation (plant, labor, railroad, and market facilities considered) can be reasonably expected to produce. The number of working places necessary to afford the assumed tonnage may be placed at 125, if mining machines are used, equivalent to an average production of 8 tons from each room and entry face.

The development account should contain two main subdivisions corresponding to class A and class B already referred to. They may be given whatever titles are considered most descriptive; perhaps Primary Development for class A and Secondary Development for class B might be deemed suitable.

Class A, or primary development, account should be charged with the actual cost (labor, materials, supervision, etc.) of hoisting and air shafts, shaft bottoms, rock slopes and tunnels, drift portals and main sumps; such grading, rock removals, masonry work and timbering as are of an unusual character on main entries designed to be used for the life of

the mine; prospecting by drill or outcrop openings. When the development period of the mine is passed, *i.e.*, when 125 working places are available, the amount in this division should be regularly offset by a depreciation credit (charged to cost of coal and based on the tonnage recoverable by the mine) of such magnitude that the entire account will be extinguished when the coal is exhausted. For example, suppose the mine under consideration had 6,900,000 tons of coal tributary to it when the development period ended and class A of the development account showed a total charge of $30,000 at that time. After 10 years of mining at the assumed annual rate, 2,300,000 tons, or one-third the original amount of coal, would have been exhausted and this account should then show a corresponding depreciation credit of $10,000.

Another plan advocated by some authorities[1] for amortizing capital invested in mining properties takes into consideration the fact that operating costs are lower, other things being equal, in the earlier years of a mine's life than toward the end when hauls are longer and maintenance charges heavier. Therefore the rate of depreciation is made largest when the mine is just developed and gradually diminished as its age increases. By this method, the depreciation credit in the above example at the end of 10 years might be $15,000. While the latter method is the better for accounting purposes, the former is preferable for appraisals.

Class B, or secondary development, account should be charged during the development period with the total actual cost of all underground work intimately connected with the recovery of coal, *i.e.*, all mining whether in narrow places or in rooms; yardage and rock work involved in the regular course of mining, such as removing 2 ft. of roof or bottom in haulageways to give sufficient head room in a 4-ft. seam; driving through clay veins and local rolls; ordinary timbering; providing usual drainage ditches; constructing brattices, overcasts and air doors. It should be credited with the estimated normal cost of the coal recovered during the development period. The debit balance would represent the net cost or value of this class of development, it being the extra expense to which the operator has been put in expanding the workings to their normal condition.

With this method of accounting, the termination of the period of development will be indicated on the books not only by the output reaching its predetermined amount but also by the actual mining cost coinciding with normal cost, and thus automatically closing the debit side of the class B account. No depreciation should be credited to this account until the entire tributary coal area has been covered by the mine workings because to maintain the normal output, sufficient development

[1] Notably Mr. J. B. L. Hornberger, Comptroller of Pittsburgh Coal Co., in a paper read before the National Coal Association, Pittsburgh, Oct. 23, 1917. Reprinted in *Coal Trade Bull.* (Nov. 1, 1917).

work must be done each year to provide new working faces to replace those worked out. So the amount and value of existing class B development remains constant until the property is nearing exhaustion.

When the entries have reached the boundaries of the mining area and only room and pillar coal remain, a depreciation credit should be started at a rate sufficient to extinguish the account when the mine is exhausted. The charge against cost of coal for this depreciation will not in crease mining costs, for it will merely replace, in whole or in part, the regular expenses of narrow work, yardage, etc. which the tonnage has hitherto borne.

Determining the Value of Mine Development

The problem of determining the value of mine development at an operation well past the development stage will be simple if the foregoing method of accounting has been employed. It can be gotten merely by adding the debit balances shown in class A and class B divisions of the development account. Should it be necessary to check these results by a physical examination of the mine or to make an entirely new appraisal, the following method is recommended.

Divide the development into two classes corresponding to A and B. Estimate the cost of work in class A, after determining dimensions of shafts, cubic yards of rock handled in extraordinary entry grading, volume of masonry in special walls or portals and such other data as have been indicated in discussing this class of work. Depreciate the total in the proportion that the exhausted tonnage bears to the original recoverable amount.

The cost of class B development can be estimated as follows: Determine, from the mine map or otherwise, the length of main and lateral entries with the air courses, crosscuts, room necks, brattices and overcasts necessary to afford working places enough to give the normal output. From these data and from the wage scale, contract prices, and yardage rates applying, the excess cost of this work over and above the normal cost of mining can be estimated with considerable accuracy. An additional allowance may be required for such expenses as timbering, ditching, and driving through clay veins or small rolls, which are of a local and variable nature and can usually be covered only by a general estimate. No depreciation will apply to the items in this class unless the mine has reached its boundaries and ceased to expand. The sum of the totals of classes A and B will fairly represent the value of mine development whether it is for the purpose of an engineering appraisal or to place on the books of the operator.

Some judgment will be required in determining the length of entries and air courses necessary to afford the normal tonnage and no hard and fast rule can be given to cover the many variations of the room-and-pillar system used in coal mining. Usually a short study of the mine map

will disclose a method of procedure, particularly if one showing the workings about the time they reached their normal output is available. It will usually be found that during the development stage parts of certain entries have had their rooms finished and even the pillars drawn back. Such portions (with their air courses, crosscuts, etc.) should be eliminated if the pillars have been drawn, or taken at one-half or one-third value if rooms have been driven up, and one-half or two-thirds of the tributary coal removed. It should be remembered that a pillar in process of being drawn affords a working face as well as a room or entry.

In the case of a mine past the development stage and producing at the normal rate, a valuable check on these estimates is obtained by noting the existing amount of entry having active rooms or pillars. This amount, increased by a sufficient length of main entry to support it, should give approximately the length of entries in the original development.

Where the operation is much behind in robbing and the map shows large areas with pillars standing, little or no value should be given to their entries for probably the value they originally had is now largely offset by the expense of cleaning falls, retimbering, and otherwise preparing them to handle the pillar coal.

In all the foregoing no mention has been made of mine track, electric conductors, and pipe lines laid in many parts of the workings, which increase in amount with the extension of the mine, and are considered by some as a part of the mine development. It is better practice to consider such items as parts of the operation's equipment and to treat them as such together with motors, pumps, mine cars, etc.

Summary

The period of a mine's development terminates when the active workings are sufficiently large to afford the normal output for which the operation is designed. The development account should show on the debit side the expenditures incurred in opening and expanding the underground workings during the development period, and on the credit side the normal cost of the coal produced during that period and a proper depreciation allowance. The value of development at a given period is the debit balance of this account; *i.e.*, the net cost or extraordinary expense of underground work incident to establishing a mine on its normal production basis, less depreciation based on the remaining recoverable coal.

DISCUSSION

T. H. Barrett, Washington, D. C. (written discussion).—In this paper the words "value," "cost," and "capital charges" have been more or less interchanged.

The word "value," as applied to mine development, has no place in

appraisals for either the engineer or the accountant for the reason that appreciation over depreciated cost should be reflected in the value of the coal, as it is a necessary adjunct and wholly dependent on this coal. For instance, a property having a 40-yr. life and on which $50,000 had been spent for development, might at the end of 5 years operation, be said to still have a development value of $50,000 or more based on replacement cost, although considerable depreciation had been sustained. This appreciation over depreciated cost is properly reflected in the value of the coal.

The method recommended for mining at cost of development, *i.e.*, excess of actual cost over normal operating cost, is open to two objections. First, for taxation purposes, Reg. 45, Revised, limits the amount to the excess of cost over receipts from minerals sold, as the government recognized that coal companies, other conditions being normal, invariably did more development in prosperous years, thus tending to stabilize the annual earnings. Second, normal operating cost, being as variable as the market price, is but a mere guess and, as mentioned by Mr. Dilworth, tends to increase as the mine grows older.

In the case of some shaft mines, particularly in Illinois and Indiana, the tonnage capacity, assuming the shaft can handle it, increases because of the increased number of working faces until the farthermost limits are reached by first mining, or until the property is 50 per cent. exhausted. Particularly is this true where the valuable surface precludes second mining, where maximum production could be reached just before the property is exhausted. A further difficulty is found in the fact that near the shaft 90 per cent. of the development cost is depreciated by subsequent haulage and 10 per cent. by extraction of coal, while at the extreme limits of the property the reverse is true.

The following method, as per example, is recommended for finding the net cost of development while the mine is in a development stage, *i.e.* before the operating conditions are realized, irrespective of capacity.

a. Total cost of rockwork, crosscuts, shafts erroneously charged to expense in one year		$50,000.00
b. Total tons produced in that year:		
Room coal	100,000	
Entry coal	100,000	200,000
c. Price received for all coal		$2.25
d. Cost per ton including rockwork		2.40
e. Cost per ton excluding rockwork		2.15
f. Excess of cost entry coal over room coal based on yardage account		0 12
g. Revised cost of entry coal, based on *b* and *f*		2.21
h. Net receipts from entry coal		4,000.00
i. Depreciation based on cost of prior development divided by prior developed tons		4,000.00
j. Net cost of development to be capitalized, $a - (h + i)$		42,000.00

Item *i* for the first year is eliminated. For any subsequent year, the factor used must take into account the per cent. of use for coal extraction of this prior development.

Where an increase in production has been made, after operating conditions are realized, as is frequently the case, the net cost to be capitalized can be proportioned as to the entry necessary to present output as against entry driven. Thus, in the foregoing case, if the proportion of entry coal to room coal were 10 per cent. to maintain present production, 80 per cent. of the $42,000 should be capitalized as representing development to increase production.

H. B. Fernald,* New York, N. Y.—The first point we should recognize is the distinction between cost and valuation. The ordinary basis of accounts should be cost; that is something definite and clear. As soon as we begin to bring into the accounts a basis of valuation, we get estimates based on some one's appraisal. There are times when it is fair and proper to use such estimates; that is recognized by Treasury regulations, as in the case of inventories for which an authorized basis is that of cost or market, whichever is lower. We also have recognition given to estimates by the Treasury Department requirements that the valuation of mining properties as of March 1, 1913, should be determined by appraisal and entered on the books. But every time a question of valuation comes up it is a matter for explanation and comment. In any event, we must have always a clear distinction between cost and valuation.

In commercial accounting, these two bases are often the same. A man who buys a lot of goods presumably has goods whose value bears some relation to their cost. But when money is spent in mine development and equipment, there is practically no relation between the value of the work done and its cost. The value of development so largely depends on and merges with the valuation of the mine itself. I am not at all sure how important it is to try to make a distinction at any time between the value of the development work and the value of the ore deposit.

If we get a total valuation of our developed deposits, we may make some statement separating the valuation of development work or of its cost, whichever basis may be used, and subtract this from the total valuation of the developed body, to get the two separate statements; one representing the value cost of development and the other representing the valuation for the orebody. But the important thing is to have no confusion between valuation and cost, and to recognize that the general basis always should be cost rather than value for development work.

The next point I want to speak of is the Treasury regulations. There has been a very important amendment to Article 222 of Treasury Regu-

* Loomis, Suffern, & Fernald, certified public accountants.

lations 45 since the system of accounts of the National Coal Association was published in 1919. This is also a modification of the regulations to which Mr. Dilworth referred. In the present regulations is provided:

All expenditures for development, rent, and royalty in excess of receipts from minerals sold, shall be charged to capital account recoverable through depletion, while the mine is in the development stage. Thereafter any development which adds value to the mineral deposit beyond the current year shall be carried as a deferred charge and apportioned and deducted as operating expense in the years to which it is applicable.

There is a similar provision with regard to plant and equipment:

All expenditures for plant and equipment shall be charged to capital account recoverable through depreciation, while the mine is in the development stage. Thereafter the cost of major items of plant and equipment shall be capitalized but the cost of minor items of equipment and plant, necessary to maintain the normal output, and the cost of replacement may be charged to current expense of operation.

There is also Article 224 of Treasury Regulations 45, 1920 edition, with its modifications, which allows for the recovery of development expenses and mining equipment through depletion, or depreciation, either in lump sum or together with the valuation of the ore extracted. There may be a single figure for depletion and depreciation, or there may be a separation into two distinct figures.

I do not think that in our accounting we want to be led astray by an attempt to adapt our accounts to Treasury Regulations, if the Treasury Regulations are not in accordance with a practical, valuable set of operating accounts. In so far as we can meet Treasury Regulations, we should carry our accounts in accordance with them. But where we have to choose between what some tax authorities in Washington have recommended as a basis that they think should be used on the tax return and what the practical man feels he needs to operate his property intelligently, I do not think there is any choice. We must have our accounts handled so that we can know what our properties are doing, and we should not abandon desirable accounting methods because of Treasury Regulations. This is particularly true because Treasury Regulations are subject to change.

The system of accounting recommended by the National Coal Association is excellent but this change in the wording of the regulations must be kept in mind. I would like also to refer to an article in the *Engineering and Mining Journal*, July 23, 1921, by H. J. C. MacDonald, on Standardized Mine Cost Statements. I liked his classification of development as representing expenditures: first, to gain access to; second, to expose; and third, to stope the metals.

I rather incline to a different distinction of these development expenditures. First, there is the preliminary development, or the development

work that must be done in the early history of the mine in order to bring it to a producing basis. Mr. Dilworth has stated that he was not particularly referring to this general class of development, which would include surface expense and all the general and administrative expenses of the development period. One cannot know what this work is going to be worth but it is expenditure on the preliminary development of the property. For this, I think, our basis is the total expenditure, less whatever income may be received during the development period. We may classify those expenditures: so much for building and equipment; so much for development specifically classed as such; so much for improvements; so much for the administrative staff; but, it is all an expenditure made for the general development of the property.

Then at some time we pass from this stage of preliminary development to one where the operating income absorbs the current operating expenses. Thereafter we have what I would term "capital development" and "deferred development charges." Capital development would cover the permanent development work, which is supposed to continue valuable for the entire life of the property. For this we should have our actual cost, which should be written off over the entire available tonnage, unless in the statement of accounts we have merged it with the valuation of the property and are writing it off in a unit charge which represents both the ore values and the mine development cost. The deferred development charges, such as Mr. Dilworth has been particularly referring to, would also be prorated over the proper tonnage. It is always a matter of serious question as to what is the proper tonnage over which such charges should be spread. It is also a most debatable question which expenditures are to be prorated and which are to be charged to current expenses.

These questions of what expenditures should be charged against development and the basis on which they should be written off against operations are not so much matters of theory as of sound, practical judgment. I am quite sure that the Treasury Department intends to have these features handled in the sound and fair and right way, and, if we have handled our development charges in such a manner, in most cases we will find that the Department will be ready to accept the basis we have used. In any event, if our accounts are carried on a sound basis such that they really reflect the status of our operations, we will not deceive ourselves as to what we are doing even if it should become necessary to make up a statement on some different basis in order to meet the requirements of the Department.

Irrespective of the ultimate disposition of development charges, we can probably meet almost any requirement of the Treasury Department if we will show, in an initial account, the expenditures made for development work. If we keep such an initial account, we may then show

on our costs sheets and on our accounts, "Development charges to be written off against the cost of production" and wipe them off each month, or we may show, "Development charges to be deferred" and spread them over a certain length of time. We will still give to the Department accounts on our books that will show what we did and how we believe the expenditures should be classified so that the Department can understand our method of accounting. Of course, our coal section has its own subdivisions and metals mines will have others; underground mines have their own problems and stripping mines have others; but if we can make the preliminary statement of expenditures, as first entered in our books, show to the Department what the expenditures were, we have overcome a serious problem—even though we differ from the Department as to how the items should be apportioned as charges against operations.

J. H. ALLPORT,* Barnesboro, Pa.—In the development of a property, everything should be capitalized until the property is brought up to the capacity for which it was designed. Any profits from the coal recovered during development you can credit as you please. Any additions to the property to maintain tonnage is a cost account; any additions to increase the tonnage is a capital account; any investment made, such as the driving of a drainage tunnel or a slope that is to be used over a period of months or years can be capitalized or entered as a deferred account, chargeable to operation during its life. Everything is a cost of production, no matter what it is, after the property has reached the capacity for which it was designed, its cost or valuation arrived at and the depletion and depreciation rates fixed. The systems of valuation used by the Treasury Department in income and excess profits taxes have no uniformity about them. I have gone over hundreds of valuations made by the Treasury Department, by engineers all over the country, by certified public accountants, by attorneys and by owners of properties and I have not found any valuations of coal properties that were arrived at by a uniform system.

J. B. DILWORTH.—Judging from the statements made, I think that unquestionably the keynote of this discussion is the non-agreement of engineers and accountants as to just what development is. Let us have a definite understanding among those in interest as to what shall be included in development accounts. As it is, almost no two companies, no two engineers are in harmony on this subject. We should reach some agreement as to just what is a proper charge to such an account, and what is properly included under development in appraisals.

R. D. HALL,† New York, N. Y.—Some of the statements made here are a little severe on those who drive excessive headings in any given

* Consulting Engineer.
† Editor, *Coal Age*.

period. The suggestion that, because development is completed, no charges should be made to capital account for the doing of special work, gives a high cost of coal if a manager decides to push his headings along.

In many places, it is customary at certain times of the year to drive an undue amount of heading and to use that heading later. When this is done, the cost of coal goes up considerably during the summer months and then comes down unduly during the winter. Shortage of orders in May often makes it necessary to put the men on heading work; and in winter, when there is a large demand for coal, to put them on room work where their efforts will have a greater productivity. Therefore, there should be some arrangement by which these charges can be put into some account that would not affect the cost of coal and could be withdrawn from that account when the mine, as it were, is being run on its fat. This is an extremely important matter, affecting everyone in the operating department, the expenses intended for future advantage being recorded as if they were for present advantage.

Moreover, it is unfortunate that, in many cases when the large, especially the original mine, development takes place over a long period of time, none of the charges bear interest. If the property were developed by bringing in contractors, the price that the company would have to pay for that contracted material would include interest to the contractor. If the company, on the other hand, does the whole work on regular company account and charges interest on it, it makes the valuation much less than if it were made by a contractor. Though it is true that during that time the mine is suffering a depreciation, this charge should not be made against the value of the development, the depreciation should rather be counted from the time when the mine is producing properly; or, more correctly, from the time when in good practice the mine ought to be in productive condition. The life of the property should then be considered not on the basis of what it would be from the time the new material was put into the property, but on how long it would last after the period in which operation really should begin to get on a working basis.

R. V. Norris,* Wilkes-Barre, Pa.—I do not think that the use, in this paper, of the so-called normal costs is sound, nor that the class B expenditures are sound. From an income-tax standpoint, it is pretty clearly determined that the cost of development must be credited with any returns during the development period. We cannot properly use an estimated value of the coal return, we must use the actual money that is paid in for any coal that may be sold during the development period. The class B proposition, holding the gangways opened to reach output as an undisturbed asset until the final mineral equal to original development is left, is not sound, the value of the property is gradually decreasing and the extent of the haulage is increasing. Sound business sense indicates

* Consulting Engineer.

that the value of such expenditures must be depleted during the life of the mine.

The proper capital charge is the whole net amount spent, including carrying charges and everything else, until the mine is in position to put out its estimated capacity—its intended capacity. Any expenditures after that capacity is reached are properly operating costs and should go as such. The mere question of distributing them through the year is a matter of accounting in the books. However, it all belongs in the cost of mining. It is not capital and you are only fooling yourself if you try to put your operating expense in with the cost of development.

EDWIN LUDLOW,* New York, N. Y.—How would you handle the question of a slope that has gone down to the limit of the engine that operates that hoist, so that it is necessary, in order to maintain the output of that mine, that a new and larger engine should be placed there to work deeper levels. Should that engine be charged to operation or accounting?

J. H. ALLPORT.—It should be charged to operation. If a property containing 5,000,000 tons of coal costs $1,000,000 to develop it to a capacity of 1000 tons a day, the amortization of the coal and of plant equipment is the million dollars. The amortization over the period of life would be the million dollars divided by the 5,000,000 tons. Any cost after that time to maintain tonnage is a cost of production. You could enter it on books as a deferred charge to operation and distribute its cost over a period of months or years, if you preferred to do so.

That, in particular, was one of the questions that was changed in the regulations of the Treasury Department. One of the first regulations was that all major items were chargeable to the cost of productions where made to maintain tonnage. They have since ruled so that they cut out locomotives, hoisting engines, and other seemingly large expenditures made within the tax year and require that they be capitalized. If they are capitalized, their amortization would be during the life of the property; but if entered under a deferred account, they would be charged to cost of production within a reasonable time. In my judgment, replacements and additions to maintain production are operating costs.

* Consulting Engineer.

Care of Rock Drills

By Howard R. Drullard,* Butte, Mont.

(Lake Superior Meeting, August, 1920)

To obtain the best results from hammer drills, close attention must be paid to two factors in drill maintenance, which are of equal importance; one is lubrication, the other is the shank.

With the exception of stoping drills, most modern rock drills require both oil and grease. The oil ports are in the lubricator, on or behind the hammer cylinder; the grease port for the lubrication of the chuck sleeve and rotating mechanism is on the chuck end. Ordinary machine oil is not adapted to rock drills; a heavier more gelatinous oil such as castor machine oil or liquid grease should be used. The lubricators should be filled once for every 12 or 14 ft. (3.5 to 4 m.) of hole drilled. Grease guns should be used to fill the port in the chuck end once a shift; a medium (No. 3) grease is well adapted to this purpose. Hard grease must not be put into the lubricators, as it will not flow through that part. Contrary to the popular belief, oiling a machine once or twice a shift does not provide sufficient lubrication; the drills should be oiled once for every 12 or 14 ft. of hole drilled.

The rotating handle of a stoping drill is an oil reservoir and is provided with a port for oiling. The rotating handle is packed with wicking, or similar material, which causes the oil to feed slowly from the handle to the other parts of the machine. Stoping drills require oil at least twice a shift; lighter oils than castor, such as Arctic Ammonia, may be used.

Drills used in shaft sinking can be oiled satisfactorily by placing a good-sized, drop, sight lubricator on the station above the sinking operations and connecting it with the air-line supplying the sinking drills. If this lubricator is properly filled and adjusted, a uniform oiling of the sinking machines will be effected without the necessity of oiling the drills individually. The grease end should be filled at the station or surface before each drilling period. The shift boss should see that the machines are greased and also that the lubricators at the station function properly. The life of the air-drill hose is somewhat shortened by this method, as oil attacks the inner tube; but as some oil is always present in the compressed air, this is not a serious objection to the method.

* District Manager, Denver Rock Drill Mfg. Co.

When operating wet drills, the water valve should always be closed before the air is shut off from the machine so that any water leaking from the water tube will be exhausted from the machine. After drilling is finished, the machines should not be carelessly thrown aside, but should be carried well back from the face and placed in a clean dry place. Many experienced drill runners stand the drifting and sinking machines chuck end up and pour a liberal quantity of oil into the chuck end. This prevents rusting if there is moisture in the machine; also, the oil finds its way into the small parts, such as the rotating mechanism.

Stoping drills are often stood up in the opposite manner, that is to say, with the chuck end down. Oil is then poured around the air-feed piston, often called the "feed bar." The oil flows down the piston into the air-feed cylinder, keeps the cup leathers soft and pliable and, if the leathers are somewhat worn, passes on into the other parts of the drill.

When machines have been in service for some time without being repaired, they should be sent to the surface to be cleaned and oiled. If this is not practicable, good results can be obtained by pouring 5 or 6 oz. of coal oil into the air hose, connecting it to the machine, and then running the drill for a minute or two. This will usually clean a drill quite thoroughly, but care must be taken to keep all lights away from the face for a few moments for, as the oil breaks up into very fine particles as it is exhausted from the machine, it forms an explosive mixture, which has been known to flash and burn the hands and face of the drill runner. After the machine has been thus cleaned, the lubricators filled, and several ounces of oil poured in the drill hose, the machine will be found to operate much more freely than before cleaning.

Drill Shanks

The method of forming drill shanks on a standard drill sharpener is very simple and quite generally understood. The shanks, however, must be accurately made and maintained to the dimensions shown in the accompanying figure, a variance of ¼ in. (6 mm.) in length, will often reduce the drilling speed of the machine 25 per cent.

Close attention must also be paid to the shape and location of the hole made to accommodate the water tube. To avoid excessive breakage of water tubes, this hole must be 5/16 in. (7.9 mm.) in diameter and punched to the depth of at least 3 in. (7.5 cm.) It must be in the center of the steel and, after punching, should be counterpunched slightly to prevent a sharp edge forming that will cut off the water tube. The shank, of course, should present a smooth striking face.

The shank, when properly formed, is hardened. Sometimes this process is not thoroughly understood. The operation is simple, involves no delicate judgment of temperatures or high mechanical skill, can be

learned by any intelligent blacksmith in a few moments, and makes a shank that will not batter, break, or damage the piston hammers of the rock drills.

The proper treatment of the shank begins in the forging. The steel must not be overheated, that is it must not approach a white heat. The work of forming the shank should begin as soon as the steel attains a bright red heat. The steel must not be allowed to "soak" in the fire, as this causes scaling; an unduly high air pressure in blowing the forge will also cause the steel to scale, and a scaled shank will not respond properly to the hardening process. After forming, the shanks should be annealed by being cooled gradually; preferably they should be covered with lime or ashes and allowed to cool.

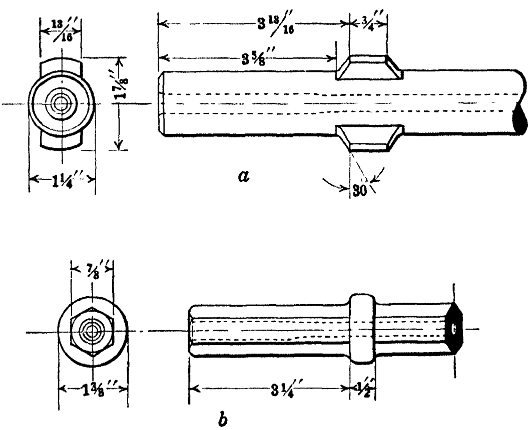


FIG. 1.—*a*. HOLLOW ROUND LUGGED DRILL STEEL, 1¼ IN. *b*. HEXAGON COLLARED DRILL STEEL, ⅞ IN.

Either fish or linseed oil is satisfactory for hardening, although other light oils are at times used. The quantity required is proportionate to the number of shanks to be hardened at one time, 5 gal. will suffice for the hardening of three or four shanks, but if considerable steel is being worked, 45 or 50 gal. are advisable.

A rectangular tank in which the steel can be stood up conveniently is generally used. A heavy screen should be placed 4 or 5 in. from the bottom in order to hold the steel above any water or dirt that may collect in the bottom of the tank. The oil must be kept free from any foreign matter, particularly water, for water collecting below the oil will overharden any hot steel that comes into contact with it.

The shank should be heated to a cherry red at the striking end with the heat graduated to a dull red just beyond the collars, or lugs; or in the case of the shankless stoping steel, to a point about 4 in. from

the end. The shank is then plunged into the oil and allowed to cool thoroughly; the operation is then finished except for testing.

One thing must be borne in mind: each and every shank must be so hardened that it can be readily cut with a file. The shanks must be softer than the piston hammers or damage to both will result. It is obvious that if two pieces of steel of approximately the same hardness are brought violently together, one or both will be damaged. If any piece cannot be readily cut with a file, it should be rejected and re-hardened and the oil tested for water.

Care must be taken that too many shanks are not treated at one time and the oil overheated; if the oil becomes hot, soft shanks will result.

The often-used draw-temper water method requires an extremely fine knowledge of color values, is seldom accurate, and is much slower than the oil method, therefore it should not be used. If the shanks are properly hardened, any well-made piston hammer will take care of itself.

Handling and Treatment of Rock-drill Steel at Copper Range Mines

By H. T. Mercer,* E. M., and A. C. Paulson,† Painesdale, Mich.

(Lake Superior Meeting, August, 1920)

The composition of steel and the theory of its heat treatment have been so ably discussed elsewhere that it is unnecessary to go into the subject here. The purpose of this paper is to describe briefly the method of treating drill steel at the Copper Range mines, giving some of the troubles encountered and the manner in which they were overcome or corrected.

To study the matter properly, it was necessary to devise a system of records that would show the actual underground performance of the drills. To illustrate this system, the records of the Champion mine are used, as it is at this mine that most of the work along these lines has been done. Reports from each drill machine are made daily by the shop and mine, and tables compiled from these reports show the footage drilled, steel broken, drills received and sent out at the shop, drills sharpened, and bits cut off. These data show what each machine or party is doing and whether the miners' supply of drills is kept up to date. There have been times when the supply of drills has been short and time lost by the miner in looking for drills. The daily records show the number and condition of drills on hand at all times.

Under the old method of sharpening, the drills were heated in a coke furnace, the proper degree of heat being judged by color. Then they were sharpened in a machine, reheated in a second coke furnace where the proper heat was again determined by color, and quenched in water. While this method was fairly successful in the hands of a careful workman and for the solid steel used in the old piston drill machine, the introduction of the faster water machine, using hollow steel, produced changes both underground and in the shop. Much more was expected of the hollow-steel and hammer-type machine, with its improved method of rotation, but troubles due to breakage, burnt bits, uneven tempering, etc., at once appeared.

* Chief Engineer, Copper Range Co.
† Assistant Master Mechanic, Champion Copper Co.

First Drill Bits Used

The drill formerly used had a cross bit, having a straight taper from the cutting edge, the ends of the wings being straight or square cornered. The cutting edge angle was 90°. With this bit as soon as the corners became worn the gage was rapidly reduced; consequently, the hole gradually decreased in diameter, causing the following drill to stick, unless a 1/8-in. (3.18 mm.) reduction in gage for every 2-ft. (0.61 m.) run of drill was maintained. This meant that the starter must have a gage of 1 3/4 in. (44.45 mm.) if the 8-ft. (2.44 m.) drill was to have a bit diameter of 1 3/8 in. (34.93 mm.). The sticking also made necessary the use of lugs, which meant more shop work.

The Carr bit was next tried. The reason for using this bit instead of applying its principle to the cross bit was that the gaging dies for shaping the ends of the wings had not at that time been perfected for the cross bit. At this time, hexagon steel was used for all single-bit drills, owing to the difficulty in centering the round steel in the gaging block as the device was then constructed.

The Carr bit made possible the use of lugless steel, owing to its free drilling qualities, because all points along the reaming edges lie in the circumference of a true circle, so that the gage is maintained even after the corners of the wings are worn. This permitted us to reduce the gage difference from 1/8 in. to 1/16 in. for the successive drills. A comparison of the old and new gage diameter is as follows:

Drill	Old, Inches	New, Inches	Reduction, Inch
Starter	1 3/4	1 1/2	1/4
4-ft. drill	1 5/8	1 7/16	3/16
6-ft. drill	1 1/2	1 3/8	1/8
8-ft. drill	1 3/8	1 5/16	1/16

From drilling experiments, it was found that a reduction of gage of 1/16 in. increased the drilling speed about 10 per cent.; therefore, by reducing the gage as shown, a total gain of about 25 per cent. was made in the drilling speed. On the other hand, the breakage of steel and the number of bits cut off in the shop was very much more for the Carr bit than for the cross bit, as shown in Table 1. This is partly due to the wear coming on a shorter cutting edge, but mostly to the difficulty in maintaining the gage and hole in the single bit. In other words, the "punishment" during the sharpening of the single bit comes mostly in one direction, which tends to flatten the hole, and the process of alternately flattening and reforming tends to straip or split the bit along the cutting edge. This tendency is not so great in the cross bit, because the sharpening blows are more evenly distributed.

The following experiment was then tried: The hole in the end of the

single bit was plugged and another hole drilled at a slight angle so as to emerge at one side in the slope or face of the bit, thus giving a continuous cutting edge. This, however, did not lessen the troubles.

TABLE 1.—*Comparison of Carr Bits and Cross Bits*

1918 Month	Carr Bit			Cross Bit		
	Number of Bits Cut off	Bits Sharpened	Per Cent. Cut off	Number of Bits Cut off	Bits Sharpened	Per Cent. Cut off
Jan	1,943	12,392	15.6	421	9,461	4.4
Feb	2,690	10,732	25.0	375	8,613	4.3
Mar	2,937	11,325	25.9	494	7,360	6.7
Apr	3,105	10,675	23.5	358	5,509	6.6
May	3,289	12,058	27.2	418	5,854	7.0
June	2,614	11,277	23.0	404	5,753	7.0
July	813	10,048	8.0	148	5,652	2.6
Aug	1,155	8,196	14 0	285	4,368	6.5
Sept	1,041	6,339	16 4	107	2,859	3.7
Oct	1,450	5,314	27.2	223	5,169	4.3
Nov				538	7,339	7.3
Dec				418	9,153	4.5
Total	21,037	98,356	21 4	4,189	77,090	5.43
Average per cent. cut off for 1919						3.98

TABLE 2.—*Consumption of Drill Steel at Champion Mine*

Year	Steel Used Pounds	Pounds Steel per Ton of Rock Broken	Total Bits Sharpened	Tons Broken per Bit Sharpened
1913	92,875	0.128	No record	
1914	144,404	0.142	No record	
1915	136,289	0.067	No record	
1916	107,684	0.054	353,421	5.67
1917	92,504	0 054	289,072	5 89
1918	63,960	0 050	177,142	7.24
1919	71,077	0 061	154,276	7.47

While the drilling speed of the single bit is somewhat greater than that of the cross bit, this is more than offset by the difficulties above mentioned. It was also found that the Carr bit was more liable to become wedged in a fissure in the rock.

Gaging blocks were finally perfected for shaping the cross bit on the Carr bit principle, viz., keeping the reaming edges on a true circle, thus allowing the same reduction in gage to be maintained, and at the same time employing the double cutting edge.

The consumption of drill steel at the Champion mine for the years 1913 to 1919, inclusive, is given in Table 2, which shows that the con-

sumption has been reduced and the efficiency of the drills increased. In 1918, the average number of bits sharpened per ton of rock broken for January was 0.172; and for December, 0.103. During 1919, a record was kept of the total rock broken and the pounds of steel per ton broken for that year are obtained from this record. Previous to 1919 no such record was kept, so that the steel per ton broken for those years had to be approximated.

The rock shipped, or stamped, cannot always be used as a basis for determining the actual tons broken, as it will be greatly affected by the degree of selection or rejection. During the later years, the rejection (also the yield) has been higher, so that the tons stamped during these years would represent a much higher tonnage broken than in the previous years. This has been taken into account in approximating the tons broken for the various years.

Dimensions of Drill Adopted

Details of the drill now used are given in Fig. 1. The shank is formed in a bulldozing machine, a pin 5 in. long being first inserted

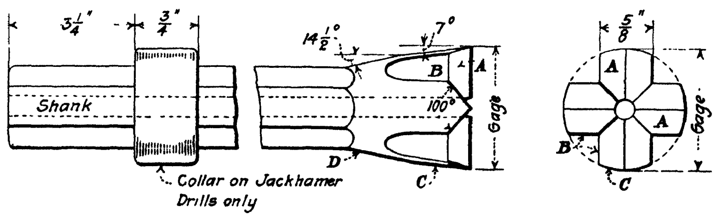


Fig. 1.—Dimensions of drills.

Name	Length as Made	⅞-in. Steel, Inches (Gage)	1-in Steel, Inches (Gage)
Starter	3 ft. 0 in.	1 7/16	1½
4-ft. drill	4 ft. 10 in.	1⅜	1 7/16
6-ft. drill	6 ft. 6 in.	1 5/16	1⅜
8-ft. drill	8 ft. 4 in.	1¼	1 5/16
10-ft. drill	10 ft. 2 in.	1 3/16	1¼

in the hole to maintain it. After the shank is formed, the pin is removed and the shank reheated and quenched in oil. A 1/64-in. (0.4 mm.) clearance is allowed between the shank and drill chuck. All shanks are made without lugs, except in a few cases where the drills are to be used in loose or "fitchering" ground. Collars are formed on the jackhammer steel only.

The bit is formed in a sharpening machine by means of a dolly, a fullering die, and a gaging block, supplemented by a gaging ring. The cutting edge and surfaces *A* are formed by the dolly, which also has a

pin for maintaining the hole. Surfaces *B* are formed by the fullering die, and the surface *C* by the gaging block. The taper *D* is shaped by the clamp that holds the drill against the blows of the dolly. The reaming edges and ends of the wings *C* are formed to a true circle; that is, if a cross-section is taken at right angles to the axis of the drill at any point along the 7° taper, the diameters in all directions will be the same. This is an application of the Carr bit principle and facilitates the drilling of true holes. Considerable trouble was experienced with "rifled" holes when using the old type of cross bit. The new type, however, has absolutely eliminated this trouble.

While working with the Carr bit, various heat-treating experiments were made. The drills were suspended in a chloride bath and the heat regulated by a pyrometer. They were then quenched in tempering oil, a 10 per cent. solution of brine, or water, and tests run on each in the mine under uniform conditions. Other tests were made with a further treatment by tempering or drawing. For example, after the hardening process the drills were again heated in oil to different degrees of temperature for drawing the temper. These experiments proved that nothing was to be gained by an additional treatment after the hardening process, and that there was no practical difference between heating in a chloride bath or in a coke furnace, provided temperature regulation was maintained equally as well in both cases. It was also found that water at a temperature of 80 to 100° F. (27 to 38° C.) made the best quenching medium for our conditions.

Practice of Champion Shop

The present practice of the Champion shop is as follows: The drills are received at the shop in iron baskets, those from each party of miners being ringed in separate bundles and each drill stamped with the serial number of the party. The baskets are lifted from the wagon or truck by an air lift supported by an overhead trolley, and run onto the sorting platform. Here they are sorted and records taken showing the number returned by each party. Drills needing repairs are placed on a rack from which they go to the repair forges. The rest are placed on the rack at the heating furnace, which is at present fired with coke, although one using oil with pyrometer control will be installed in the near future. The bits are heated to about 1900° F. (1038° C.), the proper degree of heat being judged by color, and then go to the sharpeners. Incorporated on the sharpener is a quick-acting air cylinder operating a long pin used to clear the hole in the steel, should this be found necessary.

In sharpening, care is taken to pull out the corners that have become rounded from use. The device for doing this is a part of the fullering die and consists of a pair of inclined planes set at an angle of 100°. Care

is also taken that each operation in the sharpener is not carried too far. In other words, the bit receives a few blows from the fullering die, then a few from the dolly; it then goes to the gaging block, then back to the dolly or fullering die, and so on, until the proper shape is obtained.

After sharpening, the bit is tested by the gaging ring and the drill placed on an inclined rack, which delivers it to the reheating or hardening furnace. This was formerly a coke furnace, but an oil furnace with signaling pyrometer control is now used.

This oil furnace was built at the mine from designs worked up by Mr. Paulson, to whom is due also credit for much of the work connected with the development of our present system. The construction of the furnace, with its conveyor, is illustrated in Fig. 2.

The drills are carried through the furnace by the conveyor, which delivers a drill every 35 sec., heated to a temperature of from 1450° to 1500° F. (788° to 816° C.), the heat being controlled by a Taylor signaling

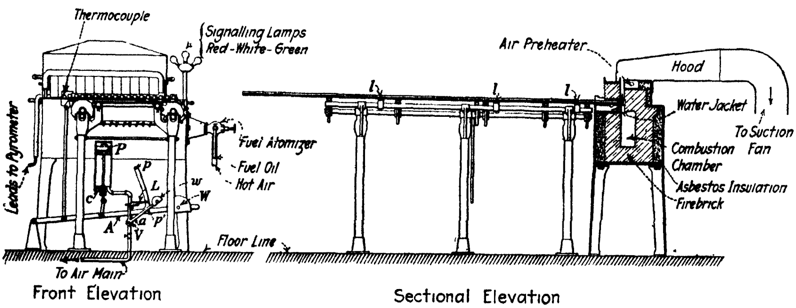


FIG. 2—TEMPERING FURNACE AND CONVEYOR.

pyrometer. The conveyor is moved by a ratchet wheel and dog, operated by an air cylinder and weight. In dropping, the large weight *W* causes the ratchet wheel to make part of a turn through a system of levers, the rate of dropping being controlled by means of the compression release cock *c*. This ratchet wheel, being keyed to the sprocket-wheel shaft, moves the conveyor chains. Upon reaching its lowest position, the pin *p* releases the latch *L*, allowing the smaller weight *w* to drop. This weight, through its arm *a*, opens the air valve *V*, admitting air to the underside of the piston *P*. This air raises the large weight *W* and its arm *A*, which in turn, by means of the pin *p′*, raises the smaller weight *w* and arm *a*, closing the air valve. The operation is then repeated. This device furnishes a simple means of operating the conveyor, the speed of which is extremely slow, without introducing a complicated system of gears or pulleys. As the drills are carried through the furnace, they are turned, or rotated, by striking the small lugs *l* on the stationary cross members of the conveyor.

The furnace confines the heat to the point of the bit and prevents its being heated too far back by having the vertical, or flame, opening narrow, and by drawing in cold air through the horizontal opening under the drills, causing it to strike the water-cooled jacket and pass up just back of the points, as shown by the arrow. The brow, or overhang, of the furnace is kept short, which also prevents the heat from being radiated too far back on the steel.

After passing through the oil furnace the bits are tested by a magnet hung on a cord to determine whether they have reached the critical, or

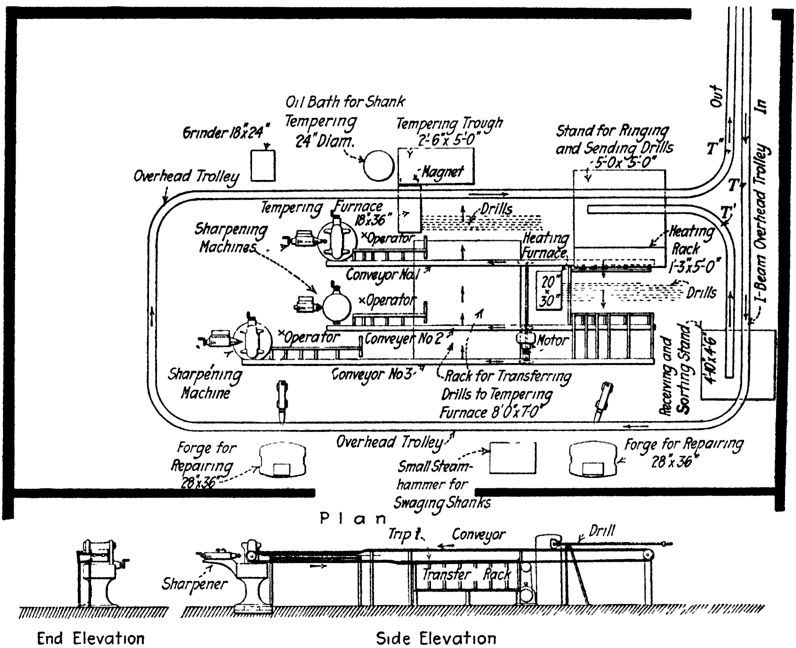


FIG. 3.—PROPOSED DRILL SHOP, CHAMPION MINE.

decalescent, point. If the magnet attracts the bit the steel has not reached the proper heat showing that the conveyor is moving too fast. The speed can easily be regulated by means of the compression release cock *c*.

There are thus two positive means of indicating whether the steel has been properly heated; the magnet test prevents underheating and the pyrometer prevents overheating. If the heat is right, the bits are quenched in a tank of water at a temperature of from 80° to 100° F. This quenching tank is a wooden box 3 ft. by 6 ft. by 2 ft. 6 in. (0.91 m. by 1.83 m. by 0.76 m.), with a false bottom, consisting of a perforated

iron plate, placed about 4 in. (10.16 cm.) from the bottom of the tank. This plate serves to keep the bits off the tank bottom, where dirt or sediment may have collected, and insures thorough quenching. While the drills are in the water, the shanks are inspected and the holes tested by a jet of air to see if they are clear. The bits are then inspected and the drills placed on the ringing stand, where they are bundled by number and returned to the mine.

At present, the drills are handled in the shop almost entirely by hand. It is intended, however, to introduce mechanical handling, with conveyors for delivering the steel to the various operators. The proposed layout is illustrated by Fig. 3. The various units are so placed with relation to one another as to reduce to a minimum all handling of the steels, thereby increasing the efficiency of the sharpening units, on which depends the capacity of the system.

The drills will be brought in by the overhead trolley *T* to the sorting platform. Those requiring repairs will be passed on by the trolley to the repair forges, and thence around to the heating rack. The rest go direct to the heating rack on the trolley *T.'* They will then be passed through an oil-fired, heating furnace by means of a conveyor, thence to the sharpeners on top of one of the three longitudinal conveyors, by which they will be delivered conveniently at the left hand of the operator. After sharpening, they are dropped, by means of an incline, on to the lower, or returning, part of the conveyor, and carried back to a trip from which, by means of an incline, they will be delivered to the conveyor at the hardening furnace. From this furnace they go to the quenching tank, thence to the ringing platform, and out by means of the trolley.

DISCUSSION

HARRISON SOUDER,* Cornwall, Pa.—We are making some experiments with the Carr, cross, and X bits; while they are not complete, the X bit seems to be doing much better work than either of the others.

H. T. MERCER.—While I think the X bit was tried, I do not know what the results were as I did not have charge of the experiments.

W. H. SCHACHT,† Painesdale, Mich.—The X bit, like the Carr bit, may have better cutting qualities than the four-point or cross bit (by that I mean a bit made with 5° taper on the wings and a cutting edge of 100°). There are, however, disadvantages inherent with the former that more than offset their better cutting qualities. This cross bit is less

* General Manager, Cornwall Ore Bank Co.
† General Manager, Copper Range Co.

susceptible to rifling, and provides a greater reaming surface at the extremity of the wings, obtaining, therefore, a greater depth of hole drilled for the same gage reduction.

In sharpening, the cross bit permits uniform and equal working of the steel from four sides; whereas with the Carr bit, most of the hammering and drawing must be done principally in one direction, which results in the splitting of the drill bit. The loss of steel due to the necessity of cutting off the split bits is considerable; we have found this to be four to five times as great as with the cross bit. The loss may not be so great with the X bit; but I believe this loss would be greater with this bit than with the cross bit. The shape of the cross bit lends itself to more uniform tempering of both cutting and reaming edges than that of the X bit, because the wings are more bulky.

Clarence M. Haight, Franklin, N. J. (written discussion).—The methods by which the records of footage drilled, steels broken, steels in and out of the shop, steels sharpened, and bits cut off were obtained would be worth describing. At one plant where this has been attempted the amount of labor involved to obtain the information was considerable.

The bits with the double taper and circular section are far superior to the old-style ones with the straight taper, and with the present mechanical sharpeners are no more trouble to form than the old ones. Has the double-arc bit[1] been tried at the Champion? In one class of ground it gave about the same results as the cross bit; it might be superior in other ground.

Since a part of the operation is to be controlled by pyrometers why not have all parts so controlled? There is a chance for an inexperienced man to burn steels on the first heat, or for a skilled man to do so should he attempt to get his heat in a shorter period of time through using a hotter furnace. If the furnace is not allowed to get above 1900° this cannot happen. With the installation described, the tendency to heat the steels for hardening before they are thoroughly cooled from forging may occur. With three machines sending drills to the hardening furnace, the drills might be put into the furnace too soon in order to prevent the steels from piling up. This results in the bit breaking off from ½ to 1 in. back from the cutting edge. This condition occurred in a shop where the steels were hardened by a second crew.

Some bits need more hammering than others; should several of these follow each other, some of the steels deposited at the sharpener by the conveyor might get too cold to work and will need reheating; apparently there is no arrangement for returning those steels to the first heating furnace.

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[1] Described by Geo. Gilman: *Eng. & Min. Jnl.* (July 28, 1917) **104**, 156.

Many shops are so situated that there is not room for conveyor systems or track systems to be installed, but where a modification of this system may be used. This modified system was installed to facilitate work in a shop where the steels were sharpened and then hardened at the same furnace and by the same men. At first, after sharpening, the steels were thrown on the floor until all of that gage were finished; they were then picked up, a few at a time, and placed in the furnace for the hardening heat. Two trucks, supporting a 30 by 60 in. (76 by 152 cm.) steel plate, horizontally, about 3 ft. above the floor, were therefore made. One of these is loaded at the dull steel racks and then placed in position at the furnace. The empty truck is placed beside the sharpener, and as the steels are finished they are placed upon it. This truck is then moved to the furnace and the first one, now empty, takes its place beside the quenching tank so that the hardened steels may be placed on it ready to be rolled to the sharp steel racks. When a separate hardening furnace was installed, only a few more stands were needed to give a very flexible system.

Relation of Air Pressure to Drilling Speeds of Hammer Drills

BY H. W. SEAMON,* JEROME, ARIZ.

(New York Meeting, February, 1921)

THE data here given were obtained by 1500 tests made by the United Verde Copper Co. to determine the most economical air pressure for the operation of hammer drills under the varying conditions of use, and to investigate the variation in drilling speed at different air pressures. Twelve models of drills were used at gage pressures ranging from 40 to 130 lb. No effort was made to harmonize theory and practice; but rather to formulate empiric rules that would cover the average variation of the results obtained. These rules on the performance of hammer drills, based on the air pressure as the main variable, however, are not necessarily of universal application, but they apparently satisfy the results obtained in this series of tests.

The drilling conditions at this property vary widely; an average of twenty-one machine shifts to a 3-ft. (0.9-m.) round has been found necessary in some of the development work, while an advance of 56 ft. has been made in seven shifts in the "oxide" ground. This range precludes the adoption of one type of drill as a standard; consequently almost every kind of hammer drill sold in this country has been tested. At present, sixteen models are in use, of which two types of the heavy (150 to 160 lb.) mounted drills, one of the light mounted drill, one stoper, and two hand plugging drills are considered as standard.

METHOD OF CONDUCTING TESTS

All machines were of the wet type, and were selected as being representative of the several types in use at this property; all had been in use from two to four months except the Waugh 66 and the Sullivan DX61, which were new. All machines were thoroughly overhauled before they were sent to the testing ground. Tests were made on the following machines: Ingersoll-Rand, No. 248 Leyner-Ingersoll, No. 18 Leyner-Ingersoll, No. 26 Baby Leyner, BCRW430 Jackhamer, and CCW11

* Mine Efficiency Engineer, United Verde Copper Co.

Stopehamer; Sullivan, DR6, DX61, DP33 Rotator, and DT44 Stoper; Waugh No. 66 Super-Dreadnaught, No. 60 Dreadnaught, and No. 21V Turbro.

The shop tests to determine the power of the drills were made on a Paynter rock drill tester, which records the number of blows per minute and the strength of blow, in foot-pounds. This machine is not an instrument of precision but it will give consistent results, if used with care and calibrated at short intervals.

The drilling was done in massive sulfide ore of uniform hardness, the same face being used for all tests. Horizontal holes were drilled with the mounted machines and vertical holes with the stopers. All tests were made with the same 40-ft. (12 m.) length of 3/4-in. (19. mm.) air-drill hose. The steel used in the tests had the following dimensions: Large mounted machines, 1 1/4-in. round; jackhammer type, 7/8-in. quarter octagon; stopers, 7/8-in. quarter octagon; length of steel, 24-in. average; type of bit, double-taper cross; diameter of bit, 1 7/8 in.; angle between cutting faces, 90°.

The higher air pressures were obtained by filling a receiver, of approximately 200 cu. ft. (5.6 cu. m.) capacity, with air at the pressure of the mains, usually 90 lb. (40 kg.) and then pumping water into the receiver until the required pressure was reached. The duration of the runs of the Paynter tester was 5 sec., with no appreciable drop in pressure; in the drilling tests, the time ranged from 10 sec. at the highest pressure to 30 sec. at 100 lb. These short runs were necessary to avoid too great a pressure drop while the test was being made and to eliminate errors caused by the rapid dulling of the steel, particularly at pressures above 100 lb. The consumption was measured with a New Jersey "Drillometer," which records the rate of flow and is, therefore, well adapted for obtaining readings at short time intervals. Special effort was made to have all the conditions uniform; the only variables were the air pressure, with the corresponding change in the number and strength of blows, and the air consumption.

It was assumed that the number of blows per minute and the strength of blow, in foot-pounds, were the same when drilling in the testing ground as the results shown by the Paynter tester under the same gage pressures. This would be true if the penetration of the drill bit in the rock and the resistance to rotation were exactly equal to the "give" in the striking piston of the tester. The figures for the air consumption of the underground tests, when corrected for the change of altitude, showed an average increase of 5 per cent. over those of the shop tests at the same pressure, indicating that there was a difference, the amount of which could not be determined. An increase in air consumption tends to increase the number of blows and the strength of blow, but in this case it was apparently required to overcome the greater resistance to rotation, a variable quan-

tity which tends to decrease the number of blows without affecting the strength of blow to any great extent. The error appears to be approximately constant and within the limits of accuracy of tests of this kind.

Analysis and Discussion of Results

For purposes of comparison, the results are shown in Tables 1 to 9. The drills are listed in three groups representing, respectively, the larger mounted machines, the jackhammer type, and the stopers.

Table 1 shows the number of blows per minute for each type of drill tested at the various pressures. Weston[1] states that the number of blows in a piston drill increases as the square root of the pressure, presumably gage pressure, and that the work done in drilling varies as the square of the velocity of the piston and this varies as the pressure. Table 1 shows that the average variation in the number of blows per minute, in terms of percentage, approximates very closely the assumption that the number of blows varies directly as the square root of the absolute pressure, rather than the gage pressure. This ratio, arbitrarily termed theoretical, is shown at the bottom of the table.

Table 1.—*Comparative Blows Per Minute*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
M1	1200	1310	1390	1465	1540	1600	1670	1738	1790	1842
M2	852	924	984	1056	1116	1176	1208	1257	1300	1356
M3	816	870	920	960	996	1032	1086	1140	1164	1188
M4	1080	1170	1250	1330	1404	1470	1510	1568	1632	1704
M5	1170	1270	1370	1455	1542	1620	1690	1754	1810	1860
M6	1400	1504	1603	1704	1798	1886	1970	2038	2088	2142
M7	1264	1368	1464	1542	1620	1720	1800	1870	1928	1968
J1	1330	1488	1704	1906	2115	2270	2410	2520	2600	2726
J2	1480	1592	1716	1875	2094	2340	2508	2632	2720	2800
J3	1296	1386	1476	1512	1596	1688	1752	1812	1848	1960
S1	1470	1610	1750	1855	1965	2044	2120	2195	2270	2320
S2	1620	1800	1955	2095	2210	2330	2435	2540	2630	2695
	Percentages, taking 80 lb. as normal									
1st group	78	84	90	95	100	105	109	113	117	121
2d group	72	78	85	92	100	108	115	120	123	129
3d group	74	82	89	95	100	105	109	114	118	120
All machines	76	82	89	94	100	106	111	115	119	122
Theoretical	75	82.2	88.5	94.4	100	105.5	110.4	115.1	119.8	124.2

[1] Eustace M. Weston: "Rock Drills, Design, Construction, and Use." N. Y., 1910. McGraw-Hill Book Co.

TABLE 2.—*Comparative Strength of Blow*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Foot-pounds per Blow									
M1	26	34	45	57	68	77	84	91	98	104
M2	22	26	29	32	36	43	48	57	65	72
M3	25	33	45	56	71	83	93	102	109	120
M4	28	46	63	78	91	101	110	117	122	125
M5	24	31	36	43	51	60	67	74	80	86
M6	23	28	33	37	40	44	49	56	67	79
M7	16½	23	31	37	43	50	56	63	66	70
J1	13½	15	16½	18	19	20½	21	21½	22	23
J2	10	12½	14	16	17	18	19½	20½	21	22
J3	12	15	18	20	22	24	27	30	33	36
S1	12	15	18	23	28	33	39	47	53	52
S2	12	17	20	23	26	29	33	37	38	34
	Percentages, taking 80 lb. as normal									
1st group	44	57	72	86	100	115	128	142	156	170
2d group	62	74	84	93	100	108	116	124	130	138
3d group ...	45	60	71	85	100	115	133	155	168	159
All machines .	49	62	75	88	100	113	125	140	151	160
Theoretical....	50	62 5	75	87.5	100	112 5	125	137.5	150	162 5

TABLE 3.—*Comparative Horsepower of Drills*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
M1	0 94	1 35	1.89	2.53	3.17	3 73	4.25	4.78	5 30	5 79
M2	0 57	0.73	0 86	1 02	1 22	1.53	1 75	2.17	2 56	2 95
M3	0.62	0 87	1.25	1 63	2 14	2 59	3.06	3 52	3 84	4 31
M4	0 92	1.63	2 38	3 14	3.87	4 50	5 02	5.55	6 03	6 45
M5	0.85	1.19	1 49	1 89	2 38	2.94	3.43	3.93	4 38	4 84
M6	0.97	1 28	1.60	1 91	2 18	2 51	2 92	3 46	4 23	5 12
M7	0.63	0.95	1 38	1 73	2 11	2.60	3.05	3.57	3.85	4.16
J1	0 54	0.63	0.85	1 04	1.22	1.41	1.53	1 64	1.73	1 90
J2	0.45	0.60	0.73	0 91	1.08	1 27	1.48	1.63	1 73	1 87
J3	0.47	0 63	0.80	0.92	1.06	1.23	1.43	1.65	1.85	2 14
S1	0.53	0.73	0.95	1.29	1.67	2.04	2 51	3.13	3.64	3 65
S2	0.59	0 93	1.18	1.46	1.74	2.05	2.44	2.85	3.03	2 78
	Percentages, taking 80 lb. as normal									
1st group......	34	49	65	82	100	120	139	161	182	204
2d group	43	57	71	85	100	117	133	147	159	177
3d group.......	33	49	63	81	100	120	145	176	196	189
All machines...	36	51	66	82	100	119	139	160	179	195
Theoretical.....	37.6	51 3	66.3	82 6	100	118.6	138	158.2	180	201 8

Table 4.—*Comparative Drilling Speed*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Inches per Minute									
M1	1.4	2.41	3.43	4.6	5.7	6.9	8.05	9.4	10.8	12.4
M7	1.42	2.1	2.84	3.48	4.24	5.2	6.1	7.18	8.4	9.6
M2	0.97	1.42	1.85	2.4	2.9	3.4	3.9	4.5	5.0	5.5
M3	1.7	2.4	3.2	3.9	4.6	5.3	6.1	6.8	7.9	9.2
M4	1.7	2.5	3.4	4.62	5.9	7.1	8.5	10.2	11.5	12.7
M5	1.62	2.3	3.1	3.95	5.0	6.15	7.4	8.6	9.8	10.95
M6	1.42	2.21	3.0	3.92	4.8	5.8	6.9	8.1	9.5	10.85
J1	0.56	1.1	1.64	2.16	2.54	2.9	3.18	3.4	3.62	3.8
J2	0.7	1.1	1.5	1.97	2.43	2.85	3.2	3.5	3.67	3.9
J3	0.75	1.1	1.55	1.9	2.23	2.65	3.1	3.4	3.7	4.0
S1	1.0	1.6	2.17	2.9	3.75	4.7	5.7	6.6	7.4	8.25
S2	1.4	2.15	2.7	3.25	3.72	4.3	5.17	5.8	6.50	6.40
	Percentages, taking 80 lb as normal									
1st group	32	47	63	81	100	120	141	165	189	214
2d group	28	46	66	84	100	117	132	144	153	163
3d group	32	50	65	82	100	121	145	166	186	196
All machines	31	47	64	82	100	119	140	160	180	198
Theoretical	37.6	51.3	66.3	82.6	100	118.6	138	158.2	180	201.8

Table 5.—*Comparative Air Consumption*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Cubic Feet of Free Air per Minute									
M1	58	74	95	113	133	150	170	189	209	227
M2	65	72	80	90	105	115	128	140	153	165
M3	58	75	85	100	112	125	140	148	156	165
M4	60	77	96	115	132	150	164	180	193	205
M5	65	86	105	125	145	165	184	203	220	237
M6	64	88	112	134	156	180	202	223	240	256
M7	60	77	92	105	120	137	148	168	182	197
J1	40	47	56	64	73	80	91	100	108	118
J2	47	56	66	77	88	100	110	121	130	140
J3	50	62	75	85	95	107	120	133	144	154
S1	35	53	65	76	84	92	101	111	120	125
S2	55	70	85	100	116	130	143	155	165	174
	Percentages, taking 80 lb. as normal									
All machines	49	62	75	87	100	113	125	137	148	156
Theoretical	50	62.5	75	87.5	100	112.5	125	137.5	150	162.5

TABLE 6.—*Comparative Efficiencies*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Per Cent									
M1	39 6	42 6	44.9	49 2	51 5	53 0	52 6	52 7	52 4	52 4
M2	21 5	24 0	24 2	24 9	25 1	28 4	28 8	32.4	34 6	36 7
M3	26 1	27.1	33 1	35 9	41 2	44.2	46 0	49 6	50 8	53 5
M4	37.5	49 5	55 7	60 1	63 3	64 0	64 5	64 4	64 5	64.5
M5	31 9	32 3	32 0	33 3	33 5	38 1	39 3	40 5	41 2	41 9
M6	37 1	34 0	32 3	31 5	30 3	29 8	30 5	32.3	36 5	41 0
M7	25 7	28 8	33 7	36 3	38 0	40 5	43 3	44 3	43 7	43.3
J1	33 1	33 6	34 1	35 7	36 1	37 6	35 4	34 2	33 1	33 0
J2	23 4	25 0	24 9	26 0	26 5	27 0	28 3	28 1	27 5	27.4
J3	23 1	23 7	24 0	23 8	24 1	24 5	25.1	25 9	26.6	28 5
S1	37 1	32 2	33.0	27 4	45 0	47.3	52 3	58 8	62 6	63 4
S2	26 2	31 0	31 3	32 2	32 4	33 7	36 0	38 4	38 0	32 8

TABLE 7.—*Comparison of Distance Drilled per Air Indicated Horsepower*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Distance, in Inches									
M1	0 59	0 76	0 81	0 89	0 93	0 97	1 0	1 04	1 07	1 12
M2	0.37	0 47	0 52	0 59	0 60	0 63	0 64	0 67	0 68	0 68
M3	0 72	0.75	0 85	0 86	0.89	0 91	0 92	0 96	1 05	1 14
M4	0 69	0.76	0 80	0 88	0 97	1.01	1 09	1 19	1 23	1 27
M5	0 61	0 62	0 67	0 70	0 75	0.80	0 85	0 89	0.92	0.95
M6	0.54	0 58	0 60	0 65	0 67	0 69	0.72	0 76	0.82	0 87
M7	0 58	0 64	0 69	0 73	0 76	0 81	0.87	0.89	0.95	1 0
J1	0 34	0 54	0 66	0 74	0 75	0 77	0 74	0.71	0.69	0 66
J2	0 37	0 46	0 51	0 56	0 60	0 61	0 61	0 60	0 58	0 57
J3	0 37	0 41	0.47	0.49	0 51	0 53	0 54	0.53	0 53	0 53
S1	0.70	0 71	0 75	0 84	0 97	1 09	1 19	1 24	1 27	1 35
S2	0 62	0.72	0.72	0 72	0.69	0 71	0.76	0 78	0 82	0 76

Table 2 shows the comparative strength of blow in the same form. There is a greater variation in the results but the average of all machines seems to indicate that the strength of blow varies directly as the gage pressure.

Table 3 shows the comparative horsepower of the several drills. Inasmuch as the horsepower is the product of the number of blows and strength of blow, divided by the constant 33,000, it may be assumed that

the variation in horsepower is as the product of the ratios shown in Tables 1 and 2, or as the gage pressure times the square root of the absolute pressure. This ratio is shown at the bottom of the table.

The comparative drilling speed is shown in Table 4. The drilling speed apparently varies directly as the power of the drill, the ratio shown in Table 3 should approximate the variation in drilling speed for the different pressures. With the exception of the lower pressures, this is the case, as is shown by the comparison of the averages for all machines and the theoretical ratio.

The comparative air consumption for the different pressures is shown in Table 5. This apparently varies directly as the gage pressure, that being the ratio shown.

Table 6 shows the relation between the horsepower of the drill, as given by the Paynter tester, and the horsepower as calculated from the air consumption and pressure by the following formula:[2]

$$W_n = 144(P_1 - P_a)V_1 \text{ ft.-lb.}$$

in which W_n = network, P_1 and P_a = absolute pressure of initial and back pressure, and V_1 = volume of compressed air. This formula gives the theoretical net work that a volume of air is capable of performing when used non-expansively. The use of the figures obtained is open to the objection that the average hammer drills used air expansively, and that the back pressure could not be recorded. This back pressure was assumed to be the same as the atmospheric pressure, 12.15 lb., the altitude being 5000 ft. The table shows the ratios, in terms of per cent., between the theoretical power, as calculated by the formula, and the observed power, as given by the tester. This may be taken as the mechanical efficiency of the drill.

Table 7 shows the relation between the power input and the measure of useful work done; that is, the rate of drilling, in inches per minute, per indicated horsepower, as calculated above. This may be considered as representing the gross efficiency, for comparison of drills of different types.

Table 8 is a comparison of the distance drilled, per horsepower, for each type of drill. This figure, which may be termed the ground factor, multiplied by the horsepower of the drill, as determined by the Paynter tester, gives the drilling speed in inches per minute. In view of the small variation under the different conditions, it would seem that the rate of drilling varies directly as the power of the drill; the rate may be expressed by the formula:

$$S = kP$$

where S = drilling speed, in inches per minute; P = power of drill as

[2] Theodore Simons: Efficiency of Compressed-Air Installations. *Eng. & Min Jnl.* (Dec. 16, 1916).

TABLE 8.—*Comparison of Distance Drilled per Machine Horsepower*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Distance, in Inches									
M1	1 49	1.79	1.81	1 82	1.80	1.85	1 90	1.97	2.04	2.14
M2	1.70	1.95	2 15	2 35	2.38	2.23	2.23	2.08	1.96	1.87
M3	2.74	2 76	2 55	2 39	2.15	2.05	2.00	1.93	2.06	2.13
M4	1.85	1.53	1.43	1 47	1.53	1 58	1.69	1.84	1.91	1.97
M5	1 91	1 93	2.08	2 09	2 10	2.09	2.16	2.19	2 24	2.27
M6	1.46	1.73	1.87	2.05	2 21	2.31	2 36	2 34	2.25	2.12
M7	2 25	2 22	2.06	2.02	2.01	2 00	2.01	2.01	2.18	2.31
J1	1.04	1 61	1 93	2 08	2 08	2.06	2.08	2.07	2.09	2.00
J2	1.55	1.83	2 06	2.16	2 25	2.25	2.16	2.15	2.12	2.12
J3	1 59	1 74	1 94	2.06	2 11	2 16	2 17	2.06	2.00	1.87
S1	1 89	2 19	2 28	2 24	2 25	2 31	2 27	2.11	2.03	2.14
S2	2 37	2 31	2 29	2 22	2 14	2.10	2 12	2.04	2 15	2.31
Average	1.82	1.96	2 04	2 08	2.08	2.08	2.09	2.07	2 09	2 10

TABLE 9.—*Comparison of Air Used per Inch Drilled*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Cubic Feet of Free Air									
M1	42.5	30.7	27 7	24 5	23 3	21 8	21.1	20 1	19.3	18 3
M2	67 0	50 6	43 4	37 5	36.2	33 8	32.9	31.1	30.6	30.0
M3	34 2	31 3	26.6	25 6	24 3	23 6	22.9	21.7	19.7	17.9
M4	35 3	30 7	28 2	24 9	22 3	21.1	19.3	17 6	16 8	16 1
M5	40.0	37.4	32.8	31 6	29 0	26 8	24.9	23.6	22.4	21.7
M6	45 0	39 8	37 3	34 2	32 4	31.1	29 3	27.5	25 2	23.6
M7	42.3	36.7	32 4	30 2	28 3	26.3	24.3	23.4	21.7	20.5
J1	71 4	42.8	34 2	29 6	28.7	27.6	28 6	29.4	29.8	31.0
J2	67.1	50.9	44 0	39 1	36 2	35.1	34 3	34.5	35.4	35.9
J3	66.5	56 4	48 4	44.7	42.5	40.4	38.7	39.1	38.9	38.5
S1	35.0	33 2	30 0	26.2	22.4	19 6	17.7	16 8	16 2	15.1
S2	39 3	32 6	31.5	30 8	31.2	30 2	27 6	26.7	25 4	27.2

given by tester for same gage pressure; and k = coefficient depending on hardness of rock and on length and gage of steel. In this series of tests, the factor k varies directly as the hardness of rock, as the length and gage of steel remained constant.

Table 9 is a comparison of the air used per inch drilled. The smaller air consumption at the higher pressures tends to compensate for the increased cost of compression so that, generally speaking, the power needed to compress the air required for drilling 1 in., or any other linear

unit, is approximately constant regardless of the pressure used. This seems to indicate that the rate of drilling varies as the power required to compress and deliver an amount of air equivalent to the con-

TABLE 10

Gage Pressure	Two-stage Compression*		$\sqrt{G}$ Ratio, Per Cent.
	Horse-power	Ratio, Per Cent.	
70	0.135	93.7	93.6
80	0.144	100.0	100.0
90	0.153	106 3	106.2
100	0.161	111.8	111.9
110	0.168	116.7	117.4
120	0.175	121.5	122.5
130	0.181	125.7	127.5

* Catalog of the Sullivan Machinery Co.

TABLE 11.—*Variation in Drilling Speed for Pressures Less Than* 120 *Lb.*

Gage Pressure	Ratio		
	$\sqrt{G^3}$, Per Cent.	Drilling Speed, Per Cent.	$G\sqrt{P}$, Per Cent.
40	35.3	31	37.6
50	49.3	47	51.3
60	65.0	64	66.3
70	82.0	82	82.6
80	100.0	100	100.0
90	119.0	119	118.6
100	140.0	140	138.0
110	161.0	160	158.2
120	183.0	180	180.0
130	207.0	198	201.8

sumption of the drill at the various gage pressures; since the rate of drilling may be expressed, conversely, as the consumption of the drill, in cubic feet, of free air per minute, divided by the amount of air required to drill 1 in. in depth.

The horsepower developed in two-stage compression, with allowance for usual losses, in compressing 1 cu. ft. of free air from atmospheric to various pressures, varies apparently as the square root of the gage pressure, as indicated in Table 10.

Since the comparative air consumption at the different pressures varies directly as the gage pressure, Table 5, and the power required for compressing 1 cu. ft. of air varies as the square root of the gage pressure, the power developed in compressing and delivering an amount of air equal to the consumption of the drill should vary directly as the product of these ratios, or as the square root of the cube of the gage pressure or as $G^{3/2}$. This ratio, the average drilling speed of all machines, and the ratio for the variation in the drilling speed as determined from the number and strength of blows, Table 4, are shown, in terms of percentage, in Table 11. This assumption more nearly approximates the variation in drilling speed for pressures less than 120 lb., than the one shown in Table 4.

Table 12 gives the comparative thermal efficiencies, or the work delivered by the drill for the quantity of air used, divided by the work required to compress and deliver the same quantity. This table is calculated on the assumption that there is a 20-lb. loss in pressure between the compressor and the working place; the pressures given being those at the drill, or the delivery pressure.

Factor of Desirability

Tillson[3] states that "in determining the relative merits of rock drills, whether of the reciprocating or hammer type, the logical basis is one of cost. Therefore, the drill which bores a foot of drill hole of standard cross-section at the lowest cost rate for drilling labor, power and mainte-

Table 12.—*Comparative Thermal Efficiencies*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
	Per Cent.									
M1	12.0	12.4	12.6	13.4	13.5	13.4	13.0	12.7	12.2	11.9
M2	6.5	6.9	6.8	6.8	6.6	7.2	7.1	7.7	8.1	8.3
M3	7.9	7.9	9.3	9.8	10.9	11.2	11.3	11.9	11.9	12.2
M4	11.3	14.4	15.7	16.3	16.7	16.2	15.9	15.4	15.1	14.7
M5	9.7	9.4	9.0	9.1	9.3	9.7	9.7	9.7	9.6	9.5
M6	11.2	9.9	9.0	8.5	7.9	7.6	7.5	7.8	8.5	9.3
M7	7.8	8.4	9.5	9.9	10.0	10.3	10.7	10.6	10.2	9.9
J1	10.0	9.8	9.6	9.7	9.5	9.5	8.7	8.2	7.7	7.5
J2	7.1	7.3	7.0	7.1	7.0	6.9	7.0	6.7	6.4	6.2
J3	6.9	6.9	6.7	6.5	6.3	6.2	6.2	6.2	6.2	6.5
S1	11.2	9.3	9.2	10.2	11.3	12.0	12.9	14.1	14.6	13.6
S2	7.9	9.0	8.8	8.7	8.5	8.5	8.9	9.2	8.9	7.5

[3] Benjamin F. Tillson: Testing and Application of Hammer Drills. *Eng. & Min Jnl.* (Apr. 7, 1917) **103,** 584.

nance (including amortization) would have the highest 'factor of desirability,' and a formula to express this may be developed as follows:

F = factor of desirability;
D = cost of drilling labor per foot of hole;
P = cost of power per foot of hole;
M = cost of maintenance per foot of hole;

then

$$F = \frac{1}{D + P + M}.\text{"}$$

The labor cost of drilling 1 ft. of hole is equal to the daily wage rate divided by the total footage drilled per shift, or rather, for our purpose,

TABLE 13.—*Factor of Desirability*

Type of Machine	Air Pressure, Pounds									
	40	50	60	70	80	90	100	110	120	130
M1	1.84	3 01	4 08	5 26	6 22	7 15	7 90	8 73	9 46	10 30
M2	1.28	1 80	2 27	2 85	3 32	3 73	4 14	4 54	4 78	5 04
M3	2.16	2 93	3 76	4 35	4 88	5 31	5 75	6 05	6 64	7 23
M4	2 16	3.05	3 96	5 10	6 13	7 00	7 86	8 92	9 45	9 72
M5	2 07	2 83	3 66	4 42	5 25	6 18	7 01	7 71	8.30	8 75
M6	1 85	2 78	3.65	4 59	5 39	6 21	7 04	7.88	8 85	9 67
M7	1 82	2.63	3 36	3 95	4 53	5 31	5 92	6.58	7.26	7 85
J1	0.75	1 46	2 15	2.79	3 22	3.61	3 84	4 06	4.23	4 35
J2	0 93	1 44	1 93	2 48	2 98	3 41	3 75	3 97	4 04	4 17
J3	1 00	1 44	1 99	2 38	2 73	3 16	3 59	3 82	4 02	4 25
S1	1 34	2 12	2 82	3 68	4 67	5 75	6 81	7 68	8 43	9 20
S2	1 83	2 72	3 32	3 86	4 23	4 70	5 39	5 79	6 32	5 86

the average footage. The footage per shift may be expressed by the relation x times the drilling speed, in inches per minute, divided by twelve; where x equals, what may be termed, the "average equivalent running time" in minutes. This factor has been determined for the several machines and the different classes of drilling, by connecting a Clark air meter to the line and observing the total amount of air used per shift, and then dividing this figure by the amount of air used per minute, as recorded by the meter, when the drill is running with full head of air. This figure is not the actual net reciprocating time, for the throttle is open when backing out of a hole and in forcing the following steel to the bottom, it is rather the average equivalent running time. This quantity has been found to average approximately 100 for all types of machines, except hand pluggers, and for every class of ground; in other words, the average shift of a miner consists of about 1½ hr. drilling.

If $x = 100$ and s = drilling speed, in inches per minute, the total footage per drill shift is $y = 100s \div 12 = 8.3s$. For a wage scale of \$6.10 per drill shift, $D = \frac{6.10}{8.3s}$ = labor cost per foot drilled.

Assuming that the cost of compressing and delivering air to the working place, at a gage pressure of 80 lb. is 5c. per 1000 cu. ft. of free air; for an interval of 100 times the normal consumption per minute, the total power cost is $0.005kV$, in which V = consumption of free air per minute, and k = coefficient depending on the pressure at which the air is used at the machine, and which varies approximately as the square root of the gage pressure. This coefficient is assumed as unity for the normal gage pressure of 80 lb.

Taking the average footage per drill shift as before, equal to $8.3s$,

$$P = \frac{0.005kV}{8.3s} = \text{power cost per foot drilled.}$$

TABLE 14.—*Results of Tests of Drilling in Porphyry*

Type of Machine	Air Pressure, Pounds	Drilling Speed, Inches per Minute	Air Consumption, Cubic Feet per Minute	Cubic Feet of Air per Inch Drilled	Distance Drilled per Air I. Hp.	Distance Drilled per Mach. Hp.
M1	40	11.3	61	5.4	4.53	12.0
	50	16.0	78	4.9	4.81	11.9
	60	21.2	100	4.7	4.79	11.2
	70	25.0	116	4.6	4.76	9.9
	80	31.0	135	4.4	4.86	9.8
	90	36.5	158	4.3	4.93	9.8
M2	40	5.4	57	10 6	2.33	9.5
	50	7.4	70	9.5	2.47	10.1
	60	9.5	83	8 7	2.59	11.0
	70	12.5	95	7 6	2.89	12.2
	80	15.0	109	7.3	2.98	12.3
	90	17.7	125	7.1	3.03	11.6
M7	40	6.2	62	10.0	2.44	9.8
	50	9.3	75	8 1	2.89	9.8
	60	12.5	90	7.2	3.14	9.1
	70	16.0	105	6 5	3.35	9.2
	80	20.2	122	6.0	3.52	9.6
	90	22.5	135	6 0	3.56	8.6
J1	40	4.2	35	8 3	2.94	7.8
	50	6.3	43	6.8	3.42	9.3
	60	8.0	53	6.6	3.40	9.4
	70	10.5	64	6.1	3.60	10.1
	80	12.5	75	6.0	3.61	10.2
	90	15.0	85	5.7	3.76	10.6

The maintenance, or upkeep cost, should increase as the pressure increases but the rate of increase could not be determined during the comparatively short time the high-pressure tests were being conducted. It is arbitrarily assumed to be directly as the square of the gage pressure; and the cost per shift is taken from the upkeep cost records.

With the same average footage per drill shift, the cost per shift for maintenance equals $k'U$, in which k' is a coefficient depending on the gage pressure and is equal to unity for a pressure of 65 lb., the average pressure at the drills, in this case; and which varies as the square of the gage pressure. U is the upkeep cost per drill shift.

Then,

$$M = \frac{k'U}{8.3s} = \text{maintenance cost per foot drilled}$$

and combining,

$$F = \frac{1}{\frac{6.10}{8.3s} + \frac{0.005kV}{8.3s} + \frac{k'U}{8.3s}} = \frac{8.3s}{6.10 + 0.005kV + k'U}$$

in which, the coefficients k and k' may be omitted where a comparison at different pressures is not desired.

The factors calculated by this formula are applicable only to the drilling conditions existing at this property. Other conditions, such as smaller air consumption at lower altitudes or a smaller upkeep cost due to easier drilling ground, would give different results and might change the comparative standing of the machines, as given in Table 13, where the factors for the several types and at the various air pressures, are shown. For the purpose of calculation, the upkeep cost of the new machines, the Waugh 66 and the Sullivan DX61, was assumed to be the same as for the 60 and the DR6, respectively.

Conclusions

1. There is little or no increase in mechanical efficiency of the drills above 90 lb. pressure; see Table 6.

2. The distance drilled per air indicated horsepower, Table 7, is greatest for the jackhamer type at 90 lb., and increases at a slow rate for the other machines at the higher pressures.

3. The average thermal efficiency is greatest at about 95 lb.

4. The factor of desirability, while increasing as the pressure, shows a comparatively slow rate of increase for pressures above 100 lb.

5. The average drill is made to be used at a pressure of 80 lb., or less; using pressures much exceeding this will invalidate the present replacement agreements with the manufacturers, thereby increasing the upkeep cost.

6. The increased breakage at the higher pressures, with the consequent greater loss of time of the drill runner in changing or repairing the machine, would tend to reduce the factor of desirability, as shown in Table 13, as this item of expense is not included therein.

7. The increased breakage of drill steel would tend to limit the pressure; although there are not sufficient data on this point to determine the maximum.

From the foregoing, it would seem that the most economical gage pressure, under the conditions obtaining at this property, is about 95 lb. at the drill.

In order to verify the conclusions drawn from these tests, additional tests were made in easier drilling ground, with a few of the machines. The conditions were the same as before, except that the pressures ranged from 40 to 90 lb., a range sufficient to demonstrate the applicability of the theoretical ratios. These results are shown in Table 14. The variation approximates closely the ratios obtained when drilling in the harder ground, which seems to indicate that the comparative performance of any type of drill, for any class of rock where the ground factor is known, and for any moderate range of air pressure can be predicted from the shop tests of the drills. The ground factors, for these two series of tests, were two and ten, respectively, as shown in Tables 8 and 14.

DISCUSSION

BENJ. F. TILLSON, Franklin Furnace, N. J.—The New Jersey drillometer is an instrument for measuring air flow by the means of multiple orifices. The meter body may be likened to a globe valve with a perforated tube mounted on the valve seat, and the valve a thin piston that can travel through this tube and uncover a variable number of small round holes. As it is designed for a certain difference in air pressure between the inlet and outlet of the meter, it functions by raising the piston a sufficient distance to permit the volume of air demanded by the tool connected to it to pass through the orifices uncovered. The indicating mechanism is a white wooden stick projecting upwards inside a glass tube on which has been calibrated a scale of air flow, in cubic feet per minute. This wooden rod is attached to the movable piston. My recollection is that there was a discrepancy of about 10 per cent. in the readings obtained with this instrument, compared to the volumetric displacement of water in metering tanks.

The Clark meter, of which Mr. Seamon later speaks, operates on the principle of the displacement of a piston. There is a reciprocating piston in the cylinder and the linear travel of the piston is recorded as the total volume of displacement. With that type of instrument as developed

some few years ago, we were unable to check results by water displacement methods I do not remember the percentage of difference involved.

Another thing that should be considered is the extreme difficulty under which the author was working to obtain true figures when he was running tests over such short periods of time as from 5 to 30 sec., as a slight discrepancy in the air pressure or of any of the quantities, such as the inches drilled, or the time interval, would cause a disproportionate error in the conclusions he reached. It would be of interest to some of us if we could study some of the individual test figures.

Breakage and Heat Treatment of Rock-drill Steel

By Benjamin F. Tillson,* E. M., Franklin, N. J.

(New York Meeting, February, 1921)

To most mine operators, it seems evident that there is a drill-steel problem, although under certain conditions the amount of drill-steel breakage does not appear serious. What is at fault? It may be one or a combination of circumstances. The development of rock drills to the hammer-drill type in place of the old reciprocating piston drill, probably is one important cause for the greater steel breakage. Perhaps the manufacturers of drill steel have failed to realize what alloys are needed for this new service or have overlooked the changes in the art of rock drilling. Although suitable alloys are provided, they may not be so handled in the manufacturing process as to be in the best condition to withstand the demands of rock drilling. The mine operator may be at fault in desiring to use the smallest possible drill-steel section in order that the gage of the drill bits may be correspondingly small and the amount of footage drilled per unit of labor may be greater; or the blacksmithshop practice at the mines may need improvement. The miner who uses the drill steel may also require more intensive supervision and education. Again, the manufacturer of rock drills may have failed to study the types of blows the steel alloys will withstand satisfactorily and which their tools are delivering and so he may not know whether or not a different design of rock drill would equal or excel their present drilling speeds without treating the drill steel so severely. All of these hypotheses probably have their supporters.

On the other hand, perhaps we are seeking too much service from drill steel and we need a fuller realization of the fatigue strains developed and should prepare to relieve these by a periodic heat treatment of the steels. However, we do not find any proof that any of these suppositions may be held responsible for the drill-steel breakage attendant to mining.

The proposed investigation of this matter by the U. S. Bureau of Mines is fully warranted in the promotion of the conservation of labor and material, and the safety of the workmen. The great range of field conditions, as well as the scope of the research, requires the energetic coöperative support of many interests. Various manufacturers of steel,

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*Assistant Superintendent, New Jersey Zinc Co.

makers of rock drills, mine operators, metallurgists, scientists, and engineering organizations (who thereby bring a coördinated interest of certain professions) must work in harmony with such agencies as the U. S. Bureau of Mines and the U. S. Bureau of Standards, whose interest would be free from any hobby or commercial bias.

In order to direct our attention to what are thought to be some of the salient features of this research, the following preliminary outline has been prepared:

U. S. Bureau of Mines Investigation of Breakage and Heat Treatment of Steel for Drilling Rock

1. Study of steels that give the greatest service before failure by breakage of drill steel: (*a*) Nature of sections; (*b*) composition; (*c*) methods of manufacture.

2. Maximum service that might be expected: (*a*) Life of steel; (*b*) methods of heat treatment; (*c*) standard methods for recording service.

3. Mechanics of failure: (*a*) Micro-analysis; (*b*) nature of stresses; (*c*) detection of incipient failure: magnetic analysis, other methods not destructive of bar of steel; (*d*) is failure due to fatigue of metal?

4. Methods and machine development for accelerated tests: (*a*) Nature of forces involved: Impact; vibratory compressional waves; combined bending, vibratory, shearing, and torsional.

5. Correlation of field tests with accelerated laboratory tests.

6. Reclamation of depreciated drill steels: (*a*) Before failure; (*b*) after failure; (*c*) methods of heating: oil-fired furnaces, electric furnaces: resistance (carbon or nichrome), induction (high-frequency or low-frequency), ohmic resistance methods; (*d*) welding of fractured steel: electric, forge.

We should also build the foundations for the necessary coöperative support to permit the speedy solution of this problem in its entirety. Many of the items that have an application to this problem have an equally or more important bearing on other arts and in themselves require considerable research. However, if we share these responsibilities and assume a definite task it will be possible for us to benefit more quickly by a completed research. For instance, the Committee on Fatigue of Metals of the Engineering Foundation has done some very important work but so far has considered chiefly non-impact repetition of stresses. The U. S. Navy Engineering Division is interested from the correlation of the impacts in drill steels to those in propeller shafts. The application of magnetic testing to a survey of the condition of stressed steel, the study of methods of welding such steel, as well as special methods of heat treatment, should prove of vital interest to

various engineering bodies and commercial interests outside of those restricted to mining; and we should hope for support from them.

Kinds of Drill Steels

Probably the so-called straight 0.90 per cent. carbon steel, of similar analysis to those usually produced in crucibles, is the most favored drill steel. The analyses are as follows:

	Per Cent.
Carbon	0.90 to 1.00
Manganese	0 10 to 0 40
Silicon	0 05 to 0.30
Sulfur	0.02 to 0.04
Phosphorus	0 005 to 0.09
Iron	the balance

Other steels, some containing vanadium, others a small amount (about 0.05 per cent.) of nickel, and carbon steels ranging, by 0.1 per cent., from 0.60 to 1.40 per cent. carbon have been tried. However, few of the tests were complete. Chrome-vanadium, chrome, and nickel steels should be experimented with; for although their initial cost may be greater their use may be justified.

Various shapes of bars have been used for drill steels. The solid octagon was quite common with the old, reciprocating-piston drills and the large cruciform was also used. With the development of the hammer drills, the solid and hollow hexagon, round, and small cruciform sections have been largely used, the dimensions varying from 7/8 to 1 1/4 in. (22 to 31.7 mm.); but the "quarter octagon" section is now fast becoming popular. This section was probably first adopted for machine drilling by the New Jersey Zinc Co. at Franklin, N. J. It had been used for hand tools by the granite workers of Vermont, who required a 1.40 per cent. carbon steel, so that steel was the first tested. The section proved to have the advantage of a longer life of drill chuck bushings, scraping the cuttings free for ejection from the drill holes, and ease of forming cross bits, but it also showed superior strength to resist the strains of service. In many instances a quarter octagon steel would give better service than much larger sections of cruciform steel. Round hollow drill steel is largely used in big shanked drifting drills, but outside of the convenience of forming the proper shanks on the round steel the hollow quarter octagon would probably prove superior.

It is important that the hole in the hollow drill steels be smooth, round, centrally located, and uniform in size or fractures will start. Any imperfections of the twist drill or punch when the ingot is pierced will remain in the bar, no matter how much it is rolled, and will provide a point for the localization of the stresses. If the temperature of the hollow bar of steel

falls too low, during hammering or rolling, the walls of the hole may show fine cracks, or cold shuts, which will prove disasterous. If any impurity or segregation of some of the constituents of the steel is concentrated, slip planes, and ultimately cleavage planes, will originate at that spot as an origin of failure and the fracture will gradually spread in an ever increasing circle until the bar parts, showing what appears as a conchoidal fracture at the break.

Life of Drill Steels

It is difficult to say what life or service should be expected from a bar of drill steel, as the conditions of field testing have many variables. However, within certain limits, we would probably agree to a minimum service. For instance, it would probably seem absurd that a 7/8-in. (22-mm.) section of steel would not drill with sharp bits over 40 ft. (12.2 m.) of 1½ in. to 1¾ in. hole in pure limestone rock, before fracturing, yet some of the accompanying graphs (Figs. 1 to 4) will indicate poorer results. The maximum drilling recorded in these tests was about 281 ft. (85.6 m.) of hole.

Such tests require a good deal of time and supervision for a drill steel does not, as a rule, drill many feet of hole in a day; it must take its turn with a number of other lengths of steel and at times is away from the work being sharpened, so that it may be under constant surveillance for many months. Unless one practically lives with such pieces of test steel, they may receive some abuse that will be responsible for their failure. If identification numbers or nicks are stamped in the steel, a premature failure will occur at such points. For some reason, it seems impossible to keep paint on the drill long enough to be worth the trouble of using it; perhaps some successful method of etching may be developed. Pasters or tags will not survive the routine mine handling.

Recently an accelerated test of a drill steel was made in limestone rock. The steel was run in a stoping drill, without having had a bit formed, and broke in 37 min. Probably it received about 1500 blows per minute or a total of about 55,500 impacts. Should we not expect a greater life than this?

With the realization that their value is doubtful, the amount of drill steel used in a large metal mine for two years and the amount of "ground" broken, shifts of drilling, feet of hole, etc., are given in Tables 1 and 2. Of course the nature of the rock or ore, the burden it is possible to blast per hole, the number of bars of drill steel kept in service compared to those used per day, the rate of drilling speed, the knowledge and industry of the drill runners, the types of rock drills used, the method of mounting or supporting the rock drills, the direction and inclination of the drill holes, the shape and condition of the drill bits when sharpened and when

TABLE 1.—*Record of Work Performed by Various Sections of Drill Steels in the Franklin Mine During* 1916–1920

Section of Drill Steel	1¼-in. Hol. Rd.	⅞-in. Q. O. Solid	⅞-in. Q. O. Hollow	1⅛-in. Hol. Rd.	All Sections
Drifts:					
Cubic feet of ground broken	1,195,138				1,195,138
Linear feet of holes drilled	788,308				788,308
Number of drill shifts	6,488				6,488
Pounds of steel used up	50,083				50,083
Average:					
Feet drilled per drill shift	121.5				121.5
Feet drilled per pound steel used up	15.7*				15.7*
Cubic feet ground broken, per pound steel used up	23.9*				23.9*
Pounds steel used up per drill shift	7.7†				7.7†
Raises:					
Cubic feet of ground broken		519,784			519,784
Linear feet of holes drilled		554,486			554,486
Number of drill shifts		4,693			4,693
Stopes:					
Cubic feet of ground broken		14,914,000			14,914,000
Linear feet of holes drilled		2,081,399	292,950		2,374,349
Number of drill shifts		17,646	3,260		20,906
Top slices:					
Cubic feet of ground broken		6,691,000			6,691,000
Linear feet of holes drilled		2,337,676	74,306		2,411,982
Number of drill shifts		24,286	930		25,216
Total:					
Cubic feet of ground broken		22,124,784			22,124,784
Linear feet of holes drilled		4,973,561	367,256		5,340,817
Number of drill shifts		46,625	4,190		50,815
Pounds of steel used up		368,728	106,774		475,502
Average:					
Feet drilled per drill shift		106.6			105.2
Feet drilled per pound steel used up		13.5†			11.2†
Cubic feet ground broken, per pound steel used up		60.0†			46.5†
Pounds steel used up per drill shift		7.9*			9.4*
Quarrying:					
Cubic feet of ground broken				9,131,002	9,131,002
Linear feet of holes drilled			186,771	629,321	816,092
Number of drill shifts			1,716	6,353	8,069
Pounds of steel used up			18,843	108,097	126,940
Average:					
Feet drilled per drill shift				99.0	101.1
Feet drilled per pound steel used up				5.8	6.4
Cubic feet ground broken, per pound steel used up				84.5	71.9
Pounds steel used up per drill shift				17.0	15.7
Grand total:					
Cubic feet of ground broken	1,195,138	22,124,784		9,131,002	32,450,924
Linear feet of holes drilled	788,308	4,973,561	554,027	629,321	6,945,217
Number of drill shifts	6,488	46,625	5.906	6,353	65,372
Pounds of steel used up	50,083	368,728	125,617	108,097	652,525
Grand average:					
Feet drilled per drill shift	121.2	106.6		99.0	106.4
Feet drilled per pound steel used up	15.7*	13.5†		5.8	10.6
Cubic feet ground broken, per pound steel used up	23.9*	60.0†		84.5	49.7
Pounds steel used up per drill shift	7.7†	7.9*		17.0	10.0

* Indicates that these figures are low because some undetermined but probably small amount of 1¼-in. round hollow steel was used in top slice development but not charged to it.

† Indicates figures are too high because of the above reasons.

TABLE 2.—*Record of Work Performed by Various Sections of Drill Steels in Sterling Hill Mine*

	1919	1920	Total 2 Years
1¼-in. Hollow Round Drill Steel, Weight 3.81 Lb per Ft.			
Drifts:			
Cubic feet ground broken	356,832	402,780	759,612
Linear feet hole drilled	213,773	227,988	441,761
Number drill shifts	1,644	1,281	2,925
Stripping:			
Cubic feet ground broken	173,733	188,998	362,731
Linear feet drilled	44,965	46,504	91,469
Number drill shifts	400	352	752
Total:			
Cubic feet broken	530,565	591,778	1,122,343
Linear feet drilled	258,738	274,492	533,230
Pounds of steel used up	13,000	10,901	23,901
Number drill shifts	2,044	1,633	3,677
Average supply of steel to drills per shift: Number of pieces and feet	(25) 130 ft.	(21) 133 ft.	(24) 131 ft.
Average:			
Feet drilled per drill shift	126.5	168.1	145.0
Feet drilled per pound steel used up	19.90	25.00	22.30
Cubic feet ground broken per pound of steel used up	40.81	54 28	46.95
Pounds steel used up per drill shift and feet	6.36 = 1.67 ft.	6.68 = 1.754 ft.	6.50 = 1.706 ft.
Per cent. depreciation of drill steel per drill shift	1.285	1.318	1.303
⅞-in. Quarter Octagon Solid Drill Steel, Weight Equals 2.47 Lb. per Ft.			
Stopes:			
Cubic feet broken	302,813	435,449	738,262
Linear feet drilled	30,484	44,439	74,923
Number drill shifts	375	584	959
Average supply of steel to drills per shift: Number of pieces and feet	(16) 84 ft.	(16) 84 ft.	(16) 84 ft.
Stripping:			
Cubic feet broken	17,376	3,947	21,323
Linear feet drilled	4,809	1,252	6,061
Number of drill shifts	52	18	70
Raising:			
Cubic feet broken	41,371	32,015	73,386
Linear feet drilled	44,168	32,784	76,952
Number drill shifts	371	248	619
Average supply of steel to drills per shift: Number of pieces and feet	(18) 79 ft.	(18) 78 ft.	(18) 79 ft.
Total:			
Cubic feet ground broken	361,560	471,411	832,971
Linear feet drilled	79,461	78,475	157,936
Pounds steel used	11,460	12,916	24,376
Number drill shifts	798	850	1,648
Average supply of steel to drills per shift: Number of pieces and feet	(16) 82 ft.	(16) 82 ft.	(16) 82 ft.
Average:			
Feet drilled per drill shift	99.6	92.3	95.8
Feet drilled per pound steel used up	6.93	6.07	6.48
Cubic feet broken per pound of steel	31.5	36.5	34.20
Pounds steel used up per drill shift and feet	14.36 = 5.82 ft.	15.20 = 6.16 ft.	14.79 = 5.982 ft.
Per cent. depreciation of drill steel per drill shift	7.10	7.51	7.30

returned as dull, the gage of the drill bits in relation to the section of the steel and force of blows impressed upon it, and various other factors influence such figures. They are, however, submitted as a stimulus to others to present similar or more precise records of the service they obtain from drill steel.

In a large mine the expense necessary to keep track of the amount of work performed in any given type of drill and rock conditions is not justified. Of course, this statement is premised by the assumption that there are many styles of drills, steels, and operating conditions in such a property. But in a small mine, where there are not so many variables, it should be possible to get accurate service records. There should, however, be some standard method for reporting such field-service tests and records.

One method of indicating the service of drill steel, given in a recent report, might lead to very deceptive conclusions. The number of broken drills are rated against the number of drills sharpened in a certain period of time. With 1¼-in. hollow steels, this percentage progressively decreased from 4.62 to 2.67 per cent., and with 1-in. hollow hexagon steels, it dropped from 0.18 to 0.06 per cent. From these figures the conclusions are drawn that the smaller section of steel gives the better service, also that there is no progressive fatigue of the metal. The early breakages are credited to initially defective steels, and when these are eliminated the endurance figures are better on the assumption that a bigger percentage of sound steel remains. It would be necessary to assume conditions about which we have no knowledge before we can give full credit to these conclusions. First, there was no increment of new steel from time to time or else it was of a superior quality. Second, the conditions of rock and rock drills must have been uniform. Third, the drill steels must have been resharpened after having drilled an equal distance in each period between sharpening, for if they had been resharpened more frequently the rate would automatically be lowered. Fourth, the type and conditions of drill bits must have been uniform. Fifth, the stock of drill steels in the mine must have been kept constant. Sixth, particular care must have been taken that drill steels that broke and were too short to be resharpened, were not thrown away in the mine and so did not appear on the records. If it is assumed that 4 per cent. broke and that after a day's drilling a steel was sharpened and reurned to work the next day, the steel would then give, on the average, only one month's service; and if drilling a 8-ft. (2.4-m.) hole dulled it, the life of the steel would then be only 200 ft. (61 m.) of drilling. If these assumptions are not greatly contrary to the true conditions, the service shown by this analysis would appear to be far less than we should hope for.

The form shown in Fig. 5 is suggested as a basis for uniform reporting of the service tests of rock-drill steels.

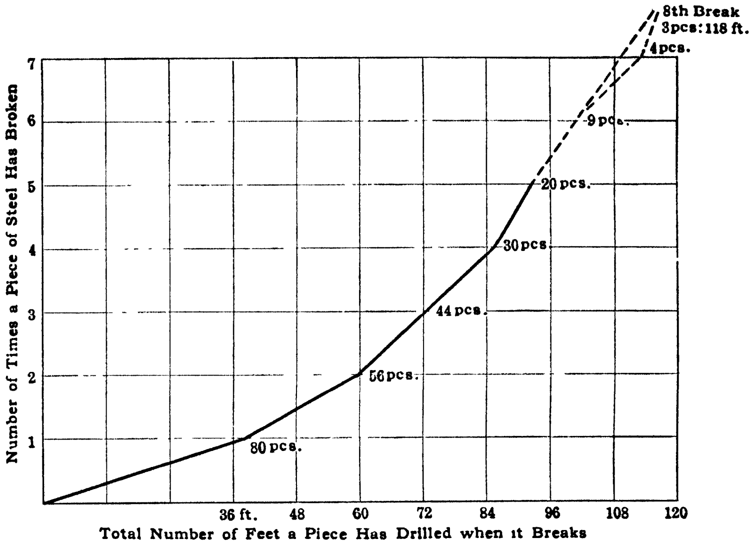


FIG. 1.—SUMMARY OF SERVICE TESTS ON SEVEN BRANDS OF ROCK DRILL STEEL. 246 PIECES OUT OF 271 BROKEN DURING TESTS; 25 PIECES AVERAGED 114 FEET WITHOUT BREAKING.

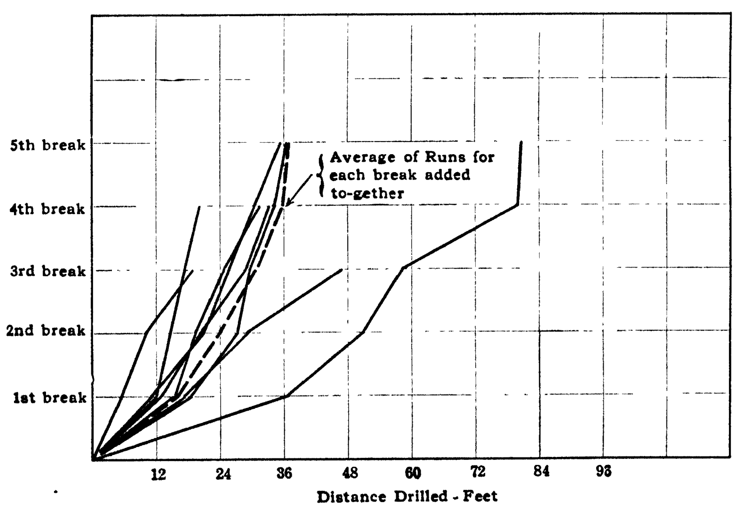


FIG. 2.—CUMULATIVE CURVES OF DISTANCES DRILLED BEFORE BREAKING BY ⅞-IN., HEXAGON, HOLLOW STEEL, BRAND E.

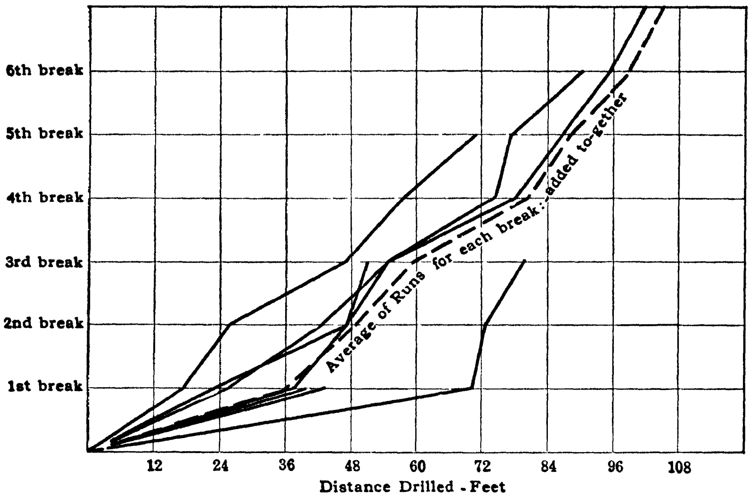


FIG. 3.—CUMULATIVE CURVES OF DISTANCES DRILLED BEFORE BREAKING BY $\frac{7}{8}$-IN., QUARTER-OCTAGON, SOLID STEEL, BRAND D.

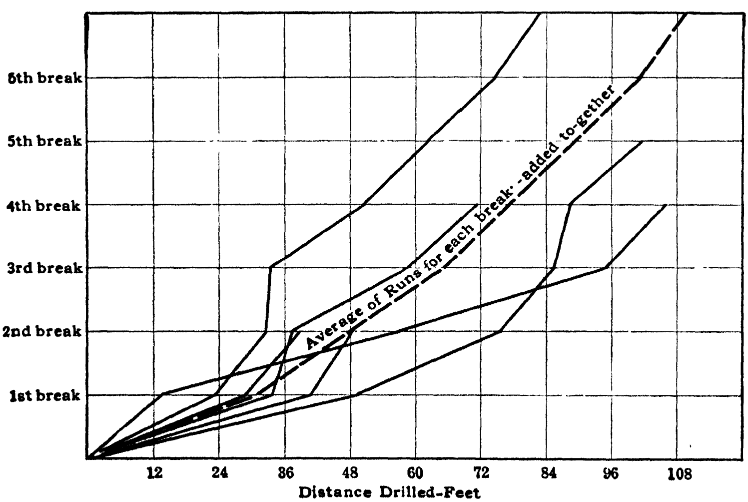


FIG. 4.—CUMULATIVE CURVES OF DISTANCES DRILLED BEFORE BREAKING BY $\frac{7}{8}$-IN., QUARTER-OCTAGON, SOLID STEEL, BRAND F.

Record of Service Test on Rock-drill Steel

Brand of Steel............................Size and Section....................
Type and Number of Rock Drill Used......Location of Tests..................
Character of Ground.....................Wet or Dry Drilling.................
Standard Drilling Speed in Ground during First Minute with 1¾-in. Gage Cross Bit..
Type of Drill Bit.........................Type of Drill Shank.................
Type of Heat Treatment of Drill Bit............of Drill Shank.................
Identification Mark for Drill-steel Test.....Length of Shank in Chuck Bushing....

Date	Length of Steel at Start of Run, in Feet and Inches	Distance Drilled in Feet and Inches	Drilling Time, in Minutes and Seconds	Average Drilling Speed, in Inches per Minute	Condition of Bit after Run	Distance of Break from Shank End	Distance of Break from Bit End	Gage of Drill Bit before Run	Gage of Drill Bit after Run	Total Times Sharpened	Total Distance Drilled to Date	Total Drilling Time to Date	Average Distance Drilled per Minute	Average Distance Drilled per Sharpening	Length of Steel Lost by Sharpening	Length of Steel Lost by Breakage	Direction of Hole	Air Pressure of Drill, in Pounds per Square Inch	Remarks

Fig. 5.

Mechanics of Failure

It is not necessary to mention the various reasons why the microscope is invaluable in considering the metallurgy of steel nor the methods involved in determining the unusual conditions in composition, grain structure, and impurities that may occur in the drill steels. For the detection of the stresses that cause incipient failure, it is necessary to prepare a polished face along the length of the entire bar of drill steel, to protect this face from oxidation with a coating such as amylacetate, and to use every care when testing this steel to destruction. Such a procedure permits the detection of the local development of slip planes that occur at the surface of the steel, but it seems difficult to detect those originating within the bar.

In one recent case microscopic analysis showed that the fracture of a drill steel was caused by an unsuspected abuse of the steel. The fractured surface had a velvety appearance, due to a martensitic structure and large crystal growth, distinct boundary lines about the crystals also showed that they had been oxidized, but the fracture occurred near the center of the bar of steel and the balance of the steel showed a pearlitic structure. A careful investigation developed the fact that, during its service, the central hole had become plugged with particles of ore, which could not be freed by hammering so the blacksmith heated the bar at the point of stoppage.

Any irregularity of form (such as an internal or external nicking or scratching), a blowhole, or a fractured zone through a crystal will produce a sudden change of section in the drill steel. The intensification of the stresses at such a point usually leads to their exceeding the elastic limit of the material. These fractures are commonly observed in drill steels, and present a conchoidal appearance because of their gradual concentric growth from a point of origin, and because of the slight motion between the two faces of a fracture plane as the compressional waves of impact traverse that section of the steel.

If there is a segregation of some physically weak compounds within the steel, it is reasonable to expect their elastic limit to be reached first and that local slip planes and cleavage planes will be developed there first. Is it not also possible that the continual transmission of longitudinal vibrations through the drill steel will tend to orient the crystals along an axis and thereby permit any cleavage planes to readily extend through one crystal to the next? May not this be responsible for the coarse-grained fractures, which are commonly referred to as crystallized steel? The appearance of large crystal faces might be supposed to be due to the extension of a cleavage plane through a number of small crystals. When slip planes have been formed, but actual cleavage planes have not been developed, will it not be possible to restore the steel to its original condition by a proper heat treatment?

If it is possible to restore steel in such condition, it is all the more important that we develop methods for detecting such incipient failure. One method of great promise is based on the detection of the stray magnetic lines of force forced out of their path through the steel by any abnormal conditions of composition or structure.

The belief that the breakage of drill steel is largely due to the so-called "fatigue" of the metal has many proponents, but others contend that the problem of fatigue is of no importance and that the steel fails through lack of a homogeneous suitable structure and that faulty manufacture or subsequent abuse are responsible. This matter should be carefully investigated, for on the conclusions will depend the necessary cures. It seems quite probable that both causes are responsible and

that any gross imperfections of the steel would tend to mask the test results, which would otherwise indicate the influence of fatigue in the failures of the metal. Reference has been made here to a published report in which it is argued that the data proves that fatigue does not enter into the breakage of drill steel. The accompanying graphs (Figs. 1 to 4) of specific tests of a certain number of bars of drill steel would seem to indicate that fatigue does have an influence. In these charts, the cumulative footage of holes drilled has been plotted as abscissas and the relative intervals at which breaks occur are plotted as ordinates. As the slopes of the lines remote from the origin are, in general, steeper than where adjacent to the origin of the chart, the drill steel became less resistant as its service was extended. There are intermediate flat slopes in some of the curves but these, probably, should be credited to the fact that the break to the left occurred prematurely because of an especially weak point in the bar. The average of these various tests for a given brand of steel, as indicated by the heavy dash line, strongly indicates a tendency for the metal to become fatigued. It would seem that the researches of the Engineering Division of the National Research Council should be extended to cover the fatigue phenomena that probably exist in drill steels, for at least two important issues are at stake. First, the development of the proper drill steel that is most resistant to fatigue and the determination of the maximum blows it can bear indefinitely without failure. Second, the application of this data to the design of rock drills so as to limit their blows within this range, and the requirement that drill steel of an adequate section be provided for use with such drills.

Forces Acting Upon Drill Steel

In order to get an idea of the magnitude of the forces acting on a drill steel, assume: that a piston weighing 4 lb. (1.8 kg.) makes 1500 strokes per minute and travels 4 in. (10.2 cm.) before it strikes the end of the drill steel; a drilling speed of 10 in. (25.4 cm.) per min. advance of the drill steel in the rock, and a cross bit of 1¾ in. gage formed on the drill steel to cut the rock and rotated halfway around in an oscillating motion 50 times per minute, the cutting edges of the drill bit are formed by planes meeting at 90° and presenting a surface 0.005 in. wide when sharp. Then the penetration of the drill steel in the rock will be about 0.1 in. per blow. The kinetic energy of the piston is about 69 ft.-lb. (9.5 m.-kg.) and the total average pressure on the rock will be about 8300 lb. (3764.8 kg.) if the compressibility of the piston, the drill steel, and the rock are ignored as relatively small factors when compared with the depth of penetration in the rock. The crushing strength of granite is estimated from 12,000 to 20,000 lb. per sq. in. (843.7 to 1406.2 kg. per sq. cm.), we will use 14,300 lb. per sq. in. If the initial pressure of contact of the steel against

the rock equals this value, the final pressure would be 16,350 lb. or 22,800 lb. per sq. in. of drill bit in contact with the rock at the end of the penetration. This is about 60 per cent. more than the pressure required to crush the rock, so a corresponding portion of the force of the blows does no useful work but would be returned as a reaction through the drill steel if both bodies were perfectly elastic. The coefficient of restoration of steel, however, is ordinarily given as about 56 per cent., so that the rock could not return quite all of this excess force as a reaction on the steel.

We must not ignore the evidence that the crushing of the rock is not alone responsible for the penetration of the drill bit, as there is also a shearing or chipping action as well as a flaking action. It would seem that the flaking action is due to the waves of rarefaction (or tension) resulting from the reflection of a part of the compressional waves delivered against the rock. When these reflected waves reach the free face of the rock, they may cause stresses exceeding the ultimate tensile strength of the rock, which thereupon liberates the portion so strained.

This same phenomenon is indicated where a projectile strikes a plate of steel. If the plate is thick enough that a hole is not punched through, a depression will be found on the side against which the force is applied and a scab of metal will be torn off and projected from the opposite side. There is, therefore, reason for believing that a considerable amount of energy is reflected back through the drill steel. If it is assumed that this steel has a modulus of elasticity equal to 29,000,000 lb. per sq. in. and a density of 489.6 lb. per cu. ft., the velocity of propagation of sound waves through it will be 16,550 ft. per sec., and as the time in which the piston is delivering its energy to the steel is 0.0005 sec., the wave lengths will be about 8.275 feet.

There seems to be some justification for the belief that a short period of contact of the piston and the drill steel produces more effective useful work on the rock than a long period and that less energy is then absorbed by the drill steel. If this is so, short wave lengths are preferable to long ones so a study of these conditions of operation might produce fruitful results for this investigation of the breakage of drill steel. Effective blows producing short waves could be obtained by designing the rock drill so that the piston was cushioned very suddenly after it had struck the drill steel, and so starting the piston on its back stroke within an extremely short time interval.

The foregoing example does not represent the true conditions existing in the operation of a hammer drill, but is an incomplete simplified analysis. The complicated nature of the stresses involved is indicated by the following description of conditions which more probably exist in practice.

When the shank end of the drill steel is struck by the piston of the

hammer drill, the other end (or bit) may be about 0.1 in. from the rock. The impact of the piston therefore produces two effects. First, a compressional wave is started through the drill steel from its shank end and will, in turn, be reflected with a change of phase from the bit end. Second, the mass of drill steel, as a whole, is given motion toward the rock; when the bit end strikes the rock, its impact starts another compressional wave from the bit end, which may in turn be reflected from the shank end with a change of phase. In addition, the natural period of vibration of the bar of steel itself imposes another series of stationary waves on the system. If, however, the frequency of the blows of the piston synchronizes with the impact waves originating at the bit end of the drill, so that the piston is in contact with the shank end of the drill steel when such waves reach that end, these waves would probably be reflected without change of phase. This last condition might indicate that the design of the rock drill and the time its piston linger on the drill steel can play an important part in the problem of the breakage of drill steel, and also greatly affect the cutting speed in the rock.

Wave Transmission

That waves are not only transmitted but are returned from the striking end and interfere so as to cause nodes of maximum stress is indicated by the fact that it is sometimes possible to detect, with the hand, points of increased temperature on a bar of steel when it is drilling ground. It seems likely that we must take into account the formation of stationary waves, and the natural periods of vibration of the various lengths and sections of drill steels may greatly influence not only the speed and efficiency of drilling but also the amount of energy that reacts on the steels.

Among the principles that govern the transmission of compressional waves are the following:

1. When a longitudinal or compressional wave passes from one medium to another, a part of the energy is reflected and a part transmitted. The phase of that part reflected is changed one-half of a wave length when the reflection occurs in the denser medium. When the reflection occurs in the lighter medium the phase of the wave is unaltered.

2. Two waves of the same length, but differing in phase, combine to produce a wave of the same length but different amplitude and phase than either of the initial waves.

3. Two waves of the same period and amplitude, but differing in phase by a wave length, combine to produce a wave of the same length but double amplitude.

4. Two waves of the same period and amplitude but differing in phase by a half wave length mutually annul each other.

5. Two waves that differ slightly in length combine to produce a wave of varying amplitude. Its frequency is one-half the sum of its component frequencies. If both initial waves were of the same ampli-

tude the amplitude of the resultant wave will vary from zero to twice the original amplitude as many times per second as the difference of their frequencies.

6. When two equal waves traverse in opposite directions, the resulting wave remains stationary. The node in the resultant wave occurs half way between the similar zero points of the original waves; and the points of greatest amplitude occur one-quarter of a wave length away. Therefore nodes occur at distances that are multiples of half wave lengths.

These principles of wave transmission must be borne in mind when we employ mechanisms for measuring the energy of the blows delivered by the rock drills, for it is usual to employ oil or other fluid to transmit the instantaneous pressures to the indicating and recording mechanism. A careful analysis should be made as to whether true readings are being obtained; an oil reservoir will influence the results by producing stationary waves of a definite maximum pressure. As an analogy, it may be likened to the effect of a condenser in an alternating-current electric transmission system. Neither would such a machine seem suitable for running the accelerated tests on drill steels, and these tests must be made if this investigation is to be pushed to a conclusion within a reasonable period of time. It will be necessary for the drill steel to bear against rock or something else of equal resilience, and methods should be devised for recording the number, force and energy of the blows delivered. Would it be possible to devise a method for measuring the compressibility of the drill steel itself while it is being tested and to compute the required data from the added information of the physical properties of the steel?

Stresses Found in Drill Steels

In addition to the compressional waves, the drill steels have to undergo bending, shearing, and torsional stresses. The bending and contingent shearing stresses are most severe in the steels used with air-feed stoping drills, for the holes are frequently drilled upwards at an angle of about 50° above the horizontal and the heavy (80 to 100 lb.) rock drill hangs upon the steel, which has about 4 in. of bearing at the end in the drill chuck while the other end is supported in the drill hole. In the hand-rotated tool, the steel is oscillated through an arc usually less than 180° and corresponding alternate bending stresses are produced. It seems quite natural that a large percentage of the breakage should occur near the point where the steel leaves the drill chuck, for the bending stresses are maximum there and the bushing may so dampen the vibrations of the steel as to cause a concentration of stresses nearby. A certain amount of torsional stress is also involved in the rotation of the drill steels, but this should be slight except where there is a tendency for the steel to get stuck in the hole. The combined stresses might be rather severe. The con-

stant rotation in one direction, which enters into the design of block-hole, sinking, and drifting drills, would seem to produce more uniform stresses; in general such drills do not punish the drill steels as much as the stoping drills. The proportion of the length of the drill steel that bears against the sides of the drill hole in the rock and receives the dampening influence of this contact also affects the stresses in the steel.

Correlation of Field Tests with Laboratory Tests

Of course the indications resulting from accelerated and laboratory tests should be corroborated by the results obtained in various mines, but it would seem wise to arrive at conclusions in the laboratory before requesting the extensive service from the mines that will be necessary, even assuming that the mines do not continue to experiment on their own initiative.

Reclamation of Depreciated Drill Steels

Although the object of this investigation is to overcome the economic loss involved in the breakage of drill steel by preventing or lessening that breakage, it would be rash to ignore the possibility that this purpose may not be obtained in the near future; we should, therefore, at the same time consider methods for reclaiming steel that has depreciated from service conditions or has been broken. There are also large quantities of short lengths of broken drill steels gathered in scrap heaps, that might be welded into serviceable drill steels. The recovery of these short pieces of drill steel might also be a considerable economic saving to the mining industry.

Assuming that magnetically, or otherwise, it is possible, on a practical operating scale, to determine that a drill steel is so fatigued that breakage is imminent, can it by suitable methods of heat treatment be returned to its original condition before it is placed in drilling service? Can the definite state of strain, or fatigue, from which it can be recovered and in excess of which its breakage is inevitable be determined? If so, what furnaces are best adapted to the heating of such steels, the length of which ordinarily will not exceed 14 ft. and on the average will be much less, and what systems of cooling or what quenching mediums should be used? Can suitable oil-fired furnaces be built for a moderate cost and can the temperatures of such furnaces be held uniform, and with what pyrometric control? The mining industry must be made acquainted with the conditions pertaining to these questions if such savings are to be hoped for.

Perhaps some type of electric furnace will be found best suited for heating long bars of steel in small quantities. If so, its design must be such as to minimize the thermal capacity and radiation of the furnace walls,

otherwise the time and electric energy required to heat the furnace in order to treat small batches of drill steel, will be disproportionate to the small number of heat units actually absorbed by the steel. Would a zone of granular or flake graphite serve as a suitable electric-resistance heating medium into which the drill steels could be plunged, or would it increase the carbon content of the steel or otherwise produce an atmosphere that would not leave the steel in proper condition?

What difficulties would be involved in the use of a cast-iron tank containing a solution of potassium carbonate as one electrode, so that when the end of the steel were plunged in the solution and the current passed through, the steel would be heated by means of its ohmic resistance? When it reached the proper temperature, the current could be shut off and the steel quenched in a separate quenching solution or it might cool in the potassium carbonate, which, of course, would then be kept at a uniform temperature by suitable cooling coils for circulating water. It would seem that such a method would provide an excellent area of contact of the steel with the electrodes so that the heating would be uniform and the distance of the heating and the temperature could be readily controlled. Of course, mechanisms could be developed for automatically performing this process and for making a variable submersion of the steel when it was being quenched so as to avoid any definite zone of demarcation between low and high temperatures.

While nichrome is generally used as the heating element in muffle furnaces, the ribbons are usually supported on the side walls so that the radiant heat does not directly impinge on the bars of steel. If the ribbons were mounted beneath an arch of proper curvature to distribute evenly the direct and reflected radiant heat waves upon the steel, better and quicker heating would result. Inductive types of electric furnaces have their places in other fields, but it may be unsafe to assume that they are equally well suited to the heat treatment of drill steel.

Perhaps they would produce local magnetic fields in the steel and cause localized eddy currents and overheating in spots; and the lack of homogeneity in the steel because of impurities, fatigue or uneven heat treatment might aggravate these effects. On the other hand, at the temperature of recalescence, local differences of temperature in the bar of steel may have been eliminated and uniformity of grain structure established. Will the use of a high-frequency alternating current establish a molecular state in the steel (by its effect on the orientation of the molecules) which will produce a superior structure or "normalize" the steel at lower temperatures than are usual? Perhaps these queries can be answered off-hand but they may need further research and then we should give them our attention.

Another electrical method of heating is to pass a current through the steel. H. P. Macdonald has commercially applied such a process to the heat treating of 0.35 per cent. carbon, 3.5 per cent. nickel steel tubes

used as automobile transmission shafts and finds that in an acceptable piece of tubing such temperature effects equalize at the critical temperature of the steel, while in a faulty piece of steel they are so exaggerated as to be an excellent index for the detection of flaws. He has consented to coöperate by investigating the suitability of this process to the heat treatment of bars of drill steel and magnetic surveys and fatigue tests are planned for the bars thus treated. The following data has been supplied by him as a result of a recent test on a bar of 7/8-in. quarter octagon solid drill steel 5½ ft. long. The bar was heated to the critical temperature in about 1 min. by passing through it an alternating, 60 cycle, electric current. It was then quenched in oil and reheated, in a similar manner, in about 2/3 min. to a temperature about 100° F. below the critical temperature and allowed to cool in air. The voltage in the primary coils of the transformer on open circuit was about 226 volts and the current in them, about in the middle of the run, was 625 amp. at 186 volts. The voltage between the clamps at the ends of the bar of drill steel was 32 volts on open circuit and 14.5 volts on closed circuit; and since the transformation ratio was about 7 to 1 and the efficiency probably 90 per cent. the average current through the steel was about 4000 amperes.

Another application of this ohmic resistance method of heating 90 per cent. carbon drill steel is the possibility of heating the ends of the steel for forging and tempering both the bit and shank ends of the steel. For this purpose an electric bolt-end heater is being adapted. A striking difference in these ohmic or inductive methods of electrical heating is that there need not be a temperature gradient from the outside of the bar to its interior and it would seem that a more uniform structure might be possible than can be obtained from external heating. Of course these points of interest in considering the reclamation of the steel also apply to the proper heat treatment of the bits and shanks, for the perfect condition of these elements is of great importance.

Electric Butt Welding of Broken Drill Steel

It is evident that where steel breakage occurs the drills are disproportionately becoming of short length and a considerable loss by scrapping will occur unless these short lengths can be welded into longer ones. In so far as solid drill steel is concerned this is not difficult, whether done by coke or oil forge heats or electrically. During the past year we thought it advisable to investigate the application of the electric butt welder for this purpose, as well as to the more difficult welding of hollow drill steel, and therefore sent pieces of various solid and hollow sections to the Thomson Electric Welding Co. Compressed air was forced through the hollow steel, while heating, in order to keep the central hole open. The welded bars had a good appearance and the holes were maintained through them. Table 3 shows the currents required and the service performed by each of these welded steels.

TABLE 3.—*Power Consumption; Machine Used, Standard 30 S. P. Welder; Primary Voltage on Open Circuit, 360 Volts*

Size of Steels	Primary Volts			Amperes			Watts			Time in Seconds			Secondary Volts Open Circuit		Total Kw.-hr.
	Preheat	Weld	Anneal	Preheat	Weld	Anneal	Preheat	Weld	Anneal	Preheat	Weld	Anneal	Preheat and Anneal	Weld	
1¼-in. hollow round	330	354	336	190	90	135	43,500	8250	28,500	10	12	20	3.5	5.5	0.306
1¼-in. hollow round	333	351	348	188	92	125	43,000	8400	22,500	11	14	20	3.5	5.5	0.288
1⅛-in. hollow round	354		356	135	35	75	18,000	3000	15,000	10	11	18	3.5	5.5	0.13
⅞-in. hollow hexagon	345		345	90	35	85	25,500	7500	22,500	10	10	17		4.6	0.196
⅞-in. hollow quarter octagon				115	45	82	18,900	9750	25,500	11	10	18	3 0	4.6	0.215
⅞-in. solid quarter octagon	342	351	354	135	35	90	28,050	6000	27,750	10	10	17	3.0	4.6	0.232

NOTE.—Not over 10 min. was required to prepare the two pieces of 1¼-in. hollow round (countersinking the ends ¼ in.) welding the same together and annealing. The ⅞-in. quarter octagon solid steel did not require over 5 min. as no countersinking was required.

7/8-in. Quarter Octagon Solid Drill Steel

An 8-ft. length was made, by two welds, from three short pieces of drill steel. It drilled 21 ft. of hole in ore without any resharpening but then broke through one of the welds. The portion carrying the remaining weld was made into a 4-ft. length so as to permit cutting off that portion near the broken weld; the steel then drilled only 4 ft. of hole before it broke about 8 in. from the shank end of the steel, but not near the weld which seemed all right. Either a Denver type 16V or an Ingersoll-Rand CC-10 stoping drill was used in this test for placing upper holes.

7/8-in. Quarter Octagon Hollow Drill Steel

Two pieces were welded to form a drill steel 7 ft. long, which:

1. Drilled 36 ft. of hole, then resharpened.
2. Drilled 81 ft. of hole, then resharpened.
3. Drilled 18 ft. of hole, was plugged by cuttings, cleaned out and resharpened.
4. Drilled 54 ft. of hole and broke about 3 in. from bit end but remote from weld; reformed bit.
5. Drilled 19 ft. of hole, then resharpened.
6. Drilled 12 ft. of hole, then resharpened.
7. Drilled 4 ft. of hole, then resharpened.
8. Drilled 34 ft. of hole, then broke but not near weld.

This steel drilled 258 ft. of hole and broke twice but did not fail in or near the weld. An Ingersoll-Rand BCR-43 block-holer was used in this test for drilling down holes in ore.

1 1/8-in. Hollow Round Drill Steel

Two pieces were welded to form a 7-ft. length:

1. Drilled 44 ft. of hole, then resharpened.
2. Drilled 40 ft. of hole and broke 3 in. from bit end but not near weld, bit reformed.
3. Drilled 48 ft. of hole, then resharpened.
4. Drilled 44 ft. of hole and broke 3 in. from the bit end but not near weld, bit reformed.
5. Drilled 51 ft. of hole, then resharpened
6. Drilled 55 ft. of hole, then resharpened
7. Drilled 46 ft. of hole, then resharpened.
8. Drilled 42 ft. of hole.

A resharpening would have caused the heated portion to so nearly approach the weld that the test was stopped. This steel drilled a total of 370 ft. of hole and broke twice but did not fail in or near the weld. An Ingersoll-Rand BCR-53 sinker drill was used in this test to drill holes in limestone.

1¼-in. Hollow Round Drill Steel

Two pieces were welded to form a 6-ft. length:

1. Drilled 4 ft. of hole, then resharpened.
2. Drilled 1½ ft. of hole, then resharpened.
3. Drilled 7 ft. of hole, then resharpened.
4. Drilled 8 ft. of hole, then resharpened.
5. Drilled 5 ft. of hole, then resharpened and broke about 8 in. away from lugs at shank end but not near weld.

The process of reshanking would have caused heating and forging so close to the weld that the test was stopped with the total drilling of 25½ ft. of hole and one break but not near the weld. A Denver Model 21 drifter was used in this test with horizontal holes in limestone and pegmatite.

These few tests indicate that it is possible electrically to butt-weld drill steel and produce a bar that is equally strong at the weld as the remaining portions of the bar. The tests were stopped as indicated in order to avoid any prejudices that might result from the breakage at a weld that had also been abused in subsequent blacksmithing. It would seem to be a proper practice to heat treat the entire bar after welding in order to assure a uniform structure throughout.

Method of Etching Steel

The following method of etching gives promise of being suitable for designating different test steels. Add four parts of glacial acetic acid to one part of absolute alcohol and let it stand about 12 hr.; then slowly and carefully add one part of nitric acid (sp. gr. 1.28). As nitrous oxide and other fumes are evolved this should not be placed in a stoppered bottle for several days, or until the reaction is completed. It may, however, be used immediately for etching steel. The surface of the steel to be etched should first be coated with a film of beeswax, tallow, or some similar substance (asphalt dissolved in chloroform has proved excellent); the symbol or number can be scratched on this coating with a sharp stylus. A low dam of beeswax should then be formed around these symbols and the etching reagent can be poured within it; 15 min. will be ample time to etch as deeply as is necessary.

Heat Treatment of Rock-drill Steel

BY GEORGE H. GILMAN, EAST BOSTON, MASS.

(New York Meeting, February, 1921)

THE campaign now being waged to improve the quality of the rock-drill bit is the natural outcome of the scientific development of the drilling machine during the past twenty years. In this development there has been a great increase of power output per unit of weight of machine. But the development of the rock drilling engine has been seriously hampered by the need of a drill bit that would withstand, without undue wear or breakage, the heavy rapid blows of the improved pneumatic hammer. This problem was difficult to solve because an increase in the weight or section of the drill-steel bar resulted in decreased cutting speed and higher operating costs of both the bit and its actuating engine. The solution, therefore, is to make the bit of steel that, for a given weight or section, will be more shock and wear resisting.

In the attempt to meet these requirements, many of the drill-steel manufacturers alloyed the steel with vanadium, chromium, nickel, etc. on the assumption that such elements would improve the rock-drill bit but due to the sensitiveness of such steel to forging and heat treatment, higher cost, and lack of continuity of effort on the part of the steel manufacturer to perfect this product, but little progress has been made in the adoption of these steels and the so-called straight carbon steel is today recognized as the standard material for all rock-drill bits.

The first step in improving drill-bit efficiency is the determination of the cause of breakage and wear. For this reason, in 1917, certain rock-drill manufacturers, in conjunction with some of the larger mining interests, conducted exhaustive tests in both the field and the laboratory. It was then found that, in the main, the cause of drill-steel troubles is not attributable to the duty imposed upon it by the normal operation of the drilling engine, but to improper forging and heat treating of the bit and to the subjecting of the bit to improper use. The fact that it is possible to minimize the troubles attributable to improper heat treatment of the drill steel has been demonstrated and it is recommended that such knowledge be made available by teaching those interested in the production, use, and up-keep of drill bits the principles on which success in this direction may be achieved.

Unfortunately, but little attention has been given to the education of the drill smith. But a few years ago, the instructions given the man

responsible for the care and up-keep of rock-drill steel were limited to such as, "Harden at a cherry red and draw temper to a straw color." Successful methods were secrets known, for the greater part, only by men who had made the work of forging and heat treating drill steel a life's task. The foregoing statement does not mean that textbooks and the records of many eminent metallurgists were not available, but that these publications were of too technical a nature for the ordinary drill-steel smith. There are, however, at present available a number of excellent books on steel and its heat treatment, a study of which will greatly assist the practical workman in acquiring a knowledge of the elements of the subject. The object of this paper is to set forth certain facts concerning the metallurgy of the present-day rock-drill steel with the hope of assisting both the smith and those seeking to improve the facilities for carrying out his work.

When steel is heated to a high temperature and allowed to cool slowly, there results a structure formed of various compositions of iron and carbon known as ferrite, cementite, and pearlite; it is the relative arrangement, amount, and size of these grains that mainly influences the physical properties of the material. Steel that has been allowed to cool slowly from a high temperature will, under the microscope, show a coarsely granular or crystalline structure, and the size of the grain is determined by the time during which it has been kept at the maximum temperature and the rate at which it is cooled. In order to refine and secure uniformity of structure, the material is worked by hammering during the period at which grain formation takes place. Steel that has been worked down to the critical temperature will show a finer grain and will be stronger than the same steel slowly cooled without being worked; it will at the same time show relatively higher ductility.

Influence of Heat Treatment in Drill Steel

The influence of heat treating on drill steel is shown in Fig. 1, which reproduces the fractured surfaces of fourteen pieces of drill steel that contained 0.85 per cent. carbon, 0.23 per cent. manganese, 0.024 per cent. phosphorus, and 0.016 per cent. sulfur. These bars, except *A* and *B* which show the fractured surface of the steel in its original state, before fracture were heated to temperatures ranging from 1200° to 2500° F. and then quenched in cold water to set the grain. Fractures *D*, *E*, and *F*, secured by quenching the pieces at a temperature of from 1400° to 1500° F., disclose a velvety structure, indicating great density and hardness, which results from the steel being quenched when its temperature was at the critical range. This condition, *D* and *E*, extends from the outer portion toward the center of the bar for about ⅛ in. (3.17 mm.), with the balance of the fracture presenting a ragged torn appearance, indicating density and toughness. The difference in the physical proper-

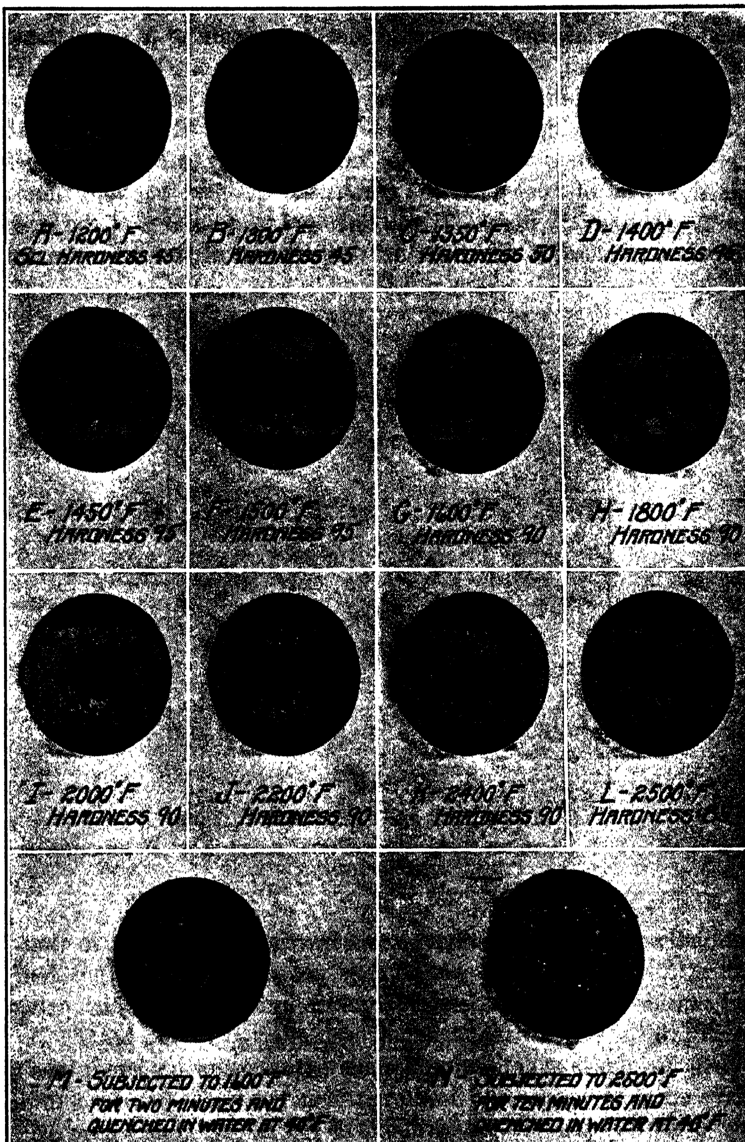


FIG. 1.—FRACTURED SPECIMENS OF 1¼ IN. ROUND HOLLOW DRILL STEELS CUT FROM THE SAME BAR. A TO L WERE HEATED TO TEMPERATURES GIVEN, HELD FOR 4 MIN. AND QUENCHED IN WATER AT 40° F.

ties of the outer and inner portions is due to the rapid cooling to which the surface of the bar was subjected; while the heat retained in the mass lowered the rate of cooling toward its center, resulting in a toughened structure of the core, which in practice affords to the cutting and reaming edges of the drill-bit shock-resisting qualities that support the more brittle but equally dense structure at the surface. The effect of the higher temperature is very apparent in fractures *F* to *L*. Fracture *M* shows the effect of quenching the bar before the heat has had time to penetrate through the mass. It will be noted that an area at the center of the piece discloses a structure identical to that of the entire fracture

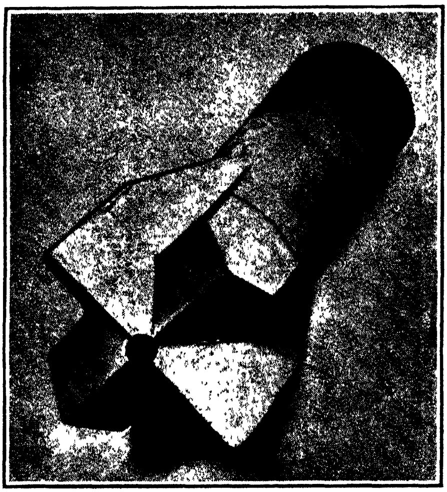


FIG. 2.—TYPICAL + DRILL BIT WITH 90° CUTTING EDGES AND 5° REAMING EDGE CLEARANCE.

B; also, that the outer portion of this fracture corresponds to that of fracture *C*, although the penetration is not so great, and that immediately inside it is the toughened structure in fracture *D* which extends inwardly to the portion that has not been affected by the heat.

With the object of determining under actual operating conditions the relative effect of proper and improper heat treatment accorded to drill bits in every day practice, five drill bits of 1⅞ in. (47.6 mm.) gage diameter were forged to the shape of the drill bit shown in Fig. 2. After being heat treated in the manner described, each was used to drill a hole 12 in. (30.5 cm.) deep in Barre granite, the time of every 2 in. (5.1 cm.) advance being observed to determine the rate of penetration throughout the depth of the drill hole. From data thus secured, curves, Fig. 3, were plotted, which graphically show the rapidity with which abnormal

wear reduces the speed of penetration. Fig. 4 shows these rock-drill bits after each had drilled the 12-in. hole. These steels contained 0.94 per cent. carbon, 0.34 per cent. manganese, 0.026 per cent. phosphorus, and 0.02 per cent. sulfur.

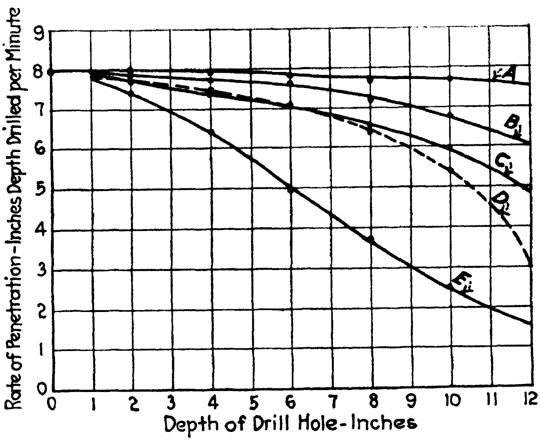
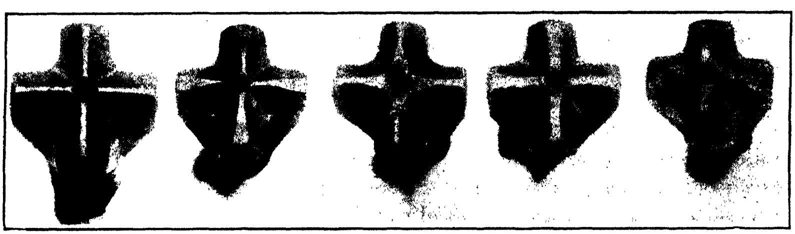


FIG. 3.—AVERAGE RESULTS FROM FIVE DRILL BITS, IDENTICAL IN MATERIAL, ORIGINAL SHAPE AND METHOD OF OPERATION, BUT VARYING IN PHYSICAL PROPERTIES DUE TO HEAT TREATMENT.



A *B* *C* *D* *E*

FIG. 4.—FIVE DRILL BITS PHOTOGRAPHED AFTER EACH HAD DRILLED A HOLE 12 IN. DEEP IN BARRE GRANITE UNDER IDENTICAL OPERATING CONDITIONS. BITS WERE OF SAME MATERIAL AND ORIGINAL SHAPE, BUT WERE SUBJECTED TO DIFFERENT HEAT TREATMENTS. *A*, FORGED AND HEAT TREATED CORRECTLY; *B*, OVERHEATED PREPARATORY TO FORGING, HEAT TREATED CORRECTLY; *C*, FORGED PROPERLY BUT HEATED TOO QUICKLY PREPARATORY TO QUENCHING; *D*, FORGED PROPERLY BUT UNDER-HEATED PREPARATORY TO QUENCHING; *E*, FORGED PROPERLY BUT OVERHEATED PREPARATORY TO QUENCHING.

Drill bit *A* was forged at a temperature of 1900° F. and allowed to cool after the completion of the forging operation. It was then heated by subjecting its end for a distance of 2 in. to a temperature of 1450° F. for 4 min. and then quenched by suddenly plunging its end to a depth of ¾ in. in cold running water, at which depth it was allowed to remain for 5 sec., and then completely cooled by slowly submerging it until after 45 sec. a depth of 4 in. was reached.

Drill bit *B* was forged at a temperature of 2400° F. and allowed to cool, after which it was subjected to the heat treatment accorded to drill bit *A*.

Drill bit *C* was forged in the same way as drill bit *A* and allowed to cool after the forging operation. Then the entire cutting and reaming edges were subjected to a temperature of 1600° F. until the color of the bit indicated a temperature of 1400° F., when the bit was removed from the furnace and quenched in the same manner as drill bit *A*.

Drill bit *D* was forged in the same way as drill bit *A* and allowed to cool after the forging operation. It was then subjected, for a distance of 2 in., to a temperature of 1350° F. for 5 min., after which it was quenched in the same manner as drill bit *A*.

Drill bit *E* was forged in the same manner as drill bit *A* and allowed to cool after the forging operation. It was then subjected, for a distance of 2 in., to a temperature of 2500° F. for 6 min. and then quenched in the same manner as drill bit *A*.

Fig. 4 shows that the variation in the wear of the gage was very pronounced. Whereas drill bit *A* was reduced in gage diameter 1/64 in. (0.397 mm.) the reduction of drill bit *E* was approximately 1/16 in.; or, in other words, four times as much as that of *A*, indicating at least sixteen times as much wear.

Factors Conducive to Drill-steel Breakage

Under ordinary working conditions there are many things conducive to drill-steel breakage. Among these are: the common practice in some districts of forging the bit with a gage diameter that, in proportion to the size of the drill-steel bar, is too great for the rock worked upon; the tendency, especially in stoping-drill practice, to allow chuck bushings to become badly worn before they are renewed; the tendency of most drill runners to operate upon drill-steel shanks that are not properly squared or with anvil blocks and hammers that have been worn concave. But the principal reason for such breakage is the practice of the drill runner to operate his machine upon a drill steel after its cutting end has been abnormally dulled in service. Then, the bit loses its ability to penetrate the rock when struck, and wear of the gage oftentimes, besides imposing a severe duty upon the rotation mechanism, subjects the bit to an abnormal torsional stress. The primary effect is reduced drilling speed, but irreparable damage may be done both to the drill steel and its engine. A dull drill bit may be caused by overheating preparatory to forging, overheating preparatory to quenching, or because it has been quenched at too low a heat, but whether or not it has been heat treated improperly, the effect of a dull bit is the same. The bit loses its ability to penetrate the rock so that the shock is absorbed by the drill steel bar or transmitted back to the actuating engine. The effect of continued

hammering on a dull bit against hard rock is similar to that of operating against an impenetrable substance.

Unfortunately, in many cases, the effect of the abuse to which the drill steel is subjected does not end with the initial fracture. When a drill steel breaks in service, the shorter end only is, as a rule, discarded and the longer piece is either reshanked or resharpened. A defective piece of drill steel is thus again put into service and, oftentimes, is again broken without being subjected to an abnormal shock. In all cases, it is essential that the rock-drill steel shall be subjected at the shank and bit ends to a heat treatment that will provide the required hardness and toughness; and as the chemical properties, especially the carbon content, vary in different steels, the heat treatment of any particular steel may best be determined by analysis of its constituents.

For the purpose of this discussion, hardness may be defined as the property of resisting penetration. With the rock-drill bit, the two forms that apply directly are abrasion hardness, which is measured by the resistance of the metal to abrasive action, and dynamic hardness, which is measured by its resistance to shock.

Primarily, the operation of hardening rock-drill steel involves heating and cooling. The function of the heating is, first, to obtain the best refinement of the grain and, second, to obtain the formation of the hardening constituents of the steel. This condition is made permanent by rapidly cooling the material, which is ordinarily accomplished by quenching in water. Steel properly hardened, when fractured, should show no trace of the original structure. Quenching, however, subjects the material to a severe test because there is set up a condition of stress which in the case of the cutting bit, is even more pronounced due to its irregular shape and non-uniformity of section. It is, therefore, essential that the operation be done correctly, for normally the defect is not detected until the steel is put in operation, and not only does the transportation of the steel from the shop to the place where it is to be used involve considerable expense and effort, but an improperly heat treated drill bit, by reason of abnormal wear or breakage of its reaming and cutting edges, may cause disaster.

Heating preparatory to hardening should be done slowly, in order to insure uniformity, and the quenching should be done at the lowest temperature that will give the desired results. Non-uniformity in heating will result in non-uniformity in cooling, which is conducive to internal strains and stresses. It is self-evident that when heating a large mass of steel, for instance, a bull bit, the temperature of the center will lag behind that at the extremities of its cutting and reaming edges. Therefore, we may assume that the temperature of the furnace at the point from which the steel is removed should, in no case, be maintained at a lower temperature than the maximum temperature to which the

drill bit itself is to be heated. A rock-drill bit heats, cools, and decarburizes at its cutting and reaming edges first, but the life of the entire bit is no greater than the life of the extremities of its wing. If the drill bit is placed in a hot furnace, the extremities are apt to be heated long before the massive part of the bit; and if the bit is removed from the furnace as soon as the desired color is apparent on the surface, the corners alone will be properly heat treated and the center of the mass will not be affected. A slow soaking heat is, therefore, necessary in order that uniformity of heat saturation may be secured and overheating of the corners avoided.

Not only is uniform heating of the steel necessary for uniform and proper hardening, but the temperature of the case, or outside of the piece, must be uniform at the moment of quenching, for a drill steel properly heated previous to its withdrawal from the furnace may have a great variation of temperature in different parts of the surface at the moment of quenching, depending on the time taken to move it from the furnace to the quenching tank and also on the influence of external draft and the manner in which it is introduced into the quenching fluid. It has been found that, in general, the less time required in moving the steel from the furnace to the quenching tank, the better will be the result secured by the quenching operation.

Forging

When forging, the steel should not be heated to a higher temperature than that which, upon being slowly reduced while the material is worked, will cause the character and size of grain to be restored to its original condition; nor should the steel be worked by hammering or squeezing after it has been cooled below the refining temperature.

When heating the drill bit preparatory to forging, it is desirable that provision be made to vary the length of the heated portion of the bar to suit conditions. For instance, when forming a new bit on the plain bar, a forging heat for from 2 to 4 in. may be required, depending on the gage and length of wing to be forged; whereas for drill bits to be resharpened, a shorter heat is desirable. It is good practice not to heat the drill bit for a greater distance than the length of the wing.

When forging the shank for forming the lugs or collar, it is desirable only to heat that portion of the bar that is to be subjected to the forging operation [according to the best practice this is a portion approximately 3 in. (7.6 cm.) long, starting from 2 to 3 in. from the tip end] instead of heating the full shank, from the tip end down, the required distance necessary to give a forging heat to the part of the bar to be upset, and then cooling off a part of the heated portion by quenching it to avoid danger of the bar end upsetting in the forging dies. The latter practice causes scoring and distortion. It has been determined that if a

forging temperature of 2000° is necessary to upset properly the collar, the bar must be heated to approximately 2300° to compensate for a drop of 300° caused by conductivity and radiation when the heated end of the shank is cooled.

In case the drill smith should inadvertently overheat the steel preparatory to forging the bit or shank end, providing the material has not been decarburized, the steel may be restored to its original condition by working it while hot and allowing it to cool slowly; but in no case should steel that has been overheated be subjected to the hardening operation before it is carefully reworked and annealed.

Heat Treatment

If the shank or cutting end is to be heat treated, it should be heated uniformly to the critical temperature and quickly quenched when the temperature is in the upper critical range, as the ultimate hardness of the piece is dependent on the rate at which it is cooled from the critical temperature. For the cutting end of the bit, it has been found to be good practice to chill quickly the cutting and reaming portions to secure the required depth of hardness and density and then to cool off the core slowly by subjecting it to a treatment less harsh. In practice this may be done by immersing, for approximately 5 sec., the cutting and reaming parts of the bit to a depth of ¾ in. (19 mm.) in running water, preferably by having the water impinge upon the surfaces to be rapidly cooled, and then gradually increasing the submergence until it is cold. This operation may best be accomplished by mechanical means, for if the temperature and timing of the operation is left to the judgment of the individual, by just so much will the finished product vary.

Overheating is the cause of most troubles that originate in the drill-steel shop. Many practical men believe that greater hardening effects may be secured by employing high temperatures; although this may be true to a limited extent, the weakening of the steel by the increase in grain size and the greater hardening strains from the high temperature more than offset the questionable production of greater hardness.

Some drill-steel smiths still believe in the efficacy of tempering drill bits "to suit the ground," claiming that for soft ground a soft drill bit is desirable and for hard ground a hard-tempered bit. Such a theory is logically wrong. While a soft drill bit may be made to give reasonably satisfactory performance in soft ground, it will fail in the harder rock; but a hard drill bit, with its cutting and reaming edges properly supported, will give good results in both soft and hard rock and, if properly heat treated, will give the best results in all kinds of rock.

The shank end should always be allowed to cool slowly after the forging operation has been performed, as this has the effect of refining the grain and removing forging strains. It should then be heated to

the critical temperature for about 7½ in. (19 cm.) from the tip end and the entire shank dipped in cyanide of potassium (crystal form), quickly quenched in oil, and allowed to remain in the oil until cool. The shank so treated is not file hard but will be dense and tough, a combination that provides shock-resisting qualities, and the increase of carbon content in the case resulting from the application of cyanide will be found to have provided wear-resisting qualities without brittleness.

Summary

Discourage the practice of operating on a dull drill bit. The salvation of rock-drill steel in service is the cushioning effect of rock penetration for every blow. Eliminate penetration, and the result is a broken drill steel.

Keep the drill-steel end of the anvil block or hammer piston ground off square and discontinue its use after it has become hollowed out.

Keep the shank end of the drill steel ground off square with the corners slightly rounded.

Keep the chuck bushing up to size as a worn bushing will tend to prevent the direct transmission of a square blow under normal operating conditions.

Subject the shank end of the drill steel to a cyanide and oil treatment in preference to water hardening.

Heat treat the cutting end of the drill bit in a manner to insure a case of maximum density and hardness and support all surfaces subject to wear with a toughened core.

Establish a maximum size of bit for every size of drill-steel bar, dependent on the character of the ground, and do not lose sight of the fact that while a bit with a gage diameter of 2½ in. (6.35 cm.) on the end of a ⅞ in. (2.2 cm.) steel bar may withstand the shock of hammering to which it is subjected when drilling in a shale rock, or clay formation, a bit of this size may be wholly unfit for drilling in a harder rock such as granite.

Analysis of Some Drill-steel Tests

BY FRANCIS B. FOLEY,* MINNEAPOLIS, MINN.

(New York Meeting, February, 1921)

WITH the possible exception of high-speed tool steel, the service demanded of rock-drill steel is probably more precarious than that of any other tool steel. Unaided by the helpful influence of alloys and dependent solely on the efficiency of the heat treatment applied for its ability to stand up under the abuse to which it is necessarily subjected, rock-drill steel is worthy of intensive study. For its heat treatment the smith has his fire, an oil fire perhaps, and a quenching bath; he judges the temperature by his eye. There are shops where excellent drills are produced by this method, but there are more where the product varies between very good and very bad. Pyrometers with specially designed equipment are used in but few cases.

But the responsibility for rock-drill failures does not rest entirely on the smith. Little has been published concerning the efficiency of drill steel, though the *Canadian Mining Journal*[1] published the breakage record of a large number of hollow drill steels. This record indicates that most of the breaks occur during the early life of the steel and that as a given batch of steel is used, its percentage of breakage decreases. This fact, the article points out, is at variance with the much-talked-of theory concerning failure from crystallization and fatigue. It seems to point to the fact that the greater part of drill-steel failure from breakage is traceable to faulty manufacture, resulting in defects in the bore of the hollow steel from inclusions of slag or oxides, etc. As this defective material meets with early failure, it is weeded out. In addition, some breakage may be caused by a lack of proper condition with respect to heat treatment of the steel as furnished by the manufacturer.

In the article mentioned, the breakage of 1¼-in. (3.17 cm.) hollow steel is given as 4.62 per cent. for the early period of use, with a decrease to 3.35 per cent. and 2.67 per cent. with time in use. For 1-in. (2.5 cm.) hexagonal hollow steel, the figures for breakage are 0.18 per cent. and 0.06 per cent.; attention is called to the great decrease in breakage by the use of smaller diameter drills. The figures are given as so many drills sharpened, but there is no way of determining the percentage of

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* Metallurgist, North Central Experiment Station, U. S. Bureau of Mines.

[1] Jan. 15, 1917.

tools broken, which must be considerably higher. For instance, from May 10 to August 9, or three months, there was a breakage of 2883 drills out of 85,882 drills sharpened, which means an average of 32 broken steels per day out of 954 steels sharpened per day. Of course, some of these broken steels are made into shorter drills and are not lost, so that it is impossible to determine, from the figures, what the true percentage of drills broken was. The figures are from records of a total of over 190,000 tools sharpened.

Breakage Caused by Faulty Practice in Forging and Hardening

Breakage near the shank and bit ends of the steel is caused, for the most part, by faulty practice in forging and hardening. The New Jersey Zinc Co., through the kindness of B. F. Tillson, has placed at the disposal of the Bureau of Mines records of drill-steel performance at Franklin, N. J. More than usual precaution was taken in the gathering of these data; those familiar with the use of drill steel will realize the troubles incident to following tools in their many trips between the blacksmith and the miner. The tests were made with different lengths of steel, practically all of them not shanked, used in different kinds of drills and tested at different times. Undoubtedly, the length of the steel, the kind of drill, and the difference in conditions existing in the testing of steel day by day have some bearing on the question of drill-steel failures. However, no steel has been consistently favored or penalized with respect to these variables. It would, of course, be ideal to have had all variables eliminated, except that of the brand of steel used, but these tests were not run for the purpose to which they have been put in this paper. Another desirable feature of this kind of testing is that all tools be tested to destruction, either until they had broken into unusable lengths or had worn too short. Records were kept of about 109 steels, representing seven brands, showing breakage, number of sharpenings, time of drilling, and footage drilled. Where breakage was recorded the pieces were measured and the position of the break thus located. All of the steels were of the same type of composition—carbon steels of 0.85 to 0.95 per cent. carbon.

As the ends of the tools are subjected to heat treatment and forging after leaving the manufacturer, only such breaks as occurred between points 10 in. from either end can safely be said to be caused by faulty manufacture. Such breaks are recorded in Table I. Brands C and E are evidently an inferior product and if they are left out, the total percentage of breakage is found to be but 4.7 per cent. This breakage was experienced in the testing of 3576 in. (90.8 m.) of drill steel and represents, numerically, three breaks, or an average of a break for each 100 ft. of steel. It can hardly be claimed that a defect for each 100 ft. of hollow drill steel is extraordinarily high; on the contrary, it seems a remarkably

TABLE 1

Brand	Number of Tools	Number of Tools that Broke	Per Cent.
A	4	0	0.0
B	4	0	0.0
C	7	3	42.8
D	11	0	0.0
E	24	14	58.3
F	6	1	16.7
G	39	2	5.1
Total	95	20	21.1

good performance and indicative of a steel of very high quality, for it is unfair to attribute all the breakage to defective steel. Granting, then, that considerable of this breakage is due to defective material, there remains a negligible number of failures to be attributed to other agencies, among which will appear so-called crystallization from fatigue. It is of interest to note that brand G is in use at the mine from which the figures appearing anonymously in the *Canadian Mining Journal* were obtained.

LOCATION OF BREAKS IN DRILL STEELS

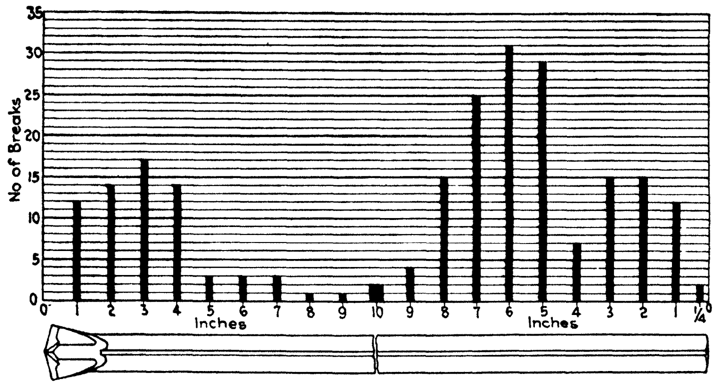
The total breakage is given in Table 2. Here a division is made of breaks that occurred within 10 in. (25.4 cm.) of the bit end, within 10 in. of the shank end, between points 10 in. from either end but nearer the bit end, and between points 10 in. from either end but nearer the shank end. There was a total of 253 breaks, of which 173, or 68.4 per cent., were in the half of the steel with the shank and 80 in the half with the bit; 68 per cent. of the breakage in the middle section of the steel was nearer the shank end than the bit end. That breakage is more prevalent in the half of the steel nearer the drill is to be expected when it is considered that, while the bit end is left free to yield, under the bending and vibratory stresses applied, the shank end is held fairly rigid in the drill. Table 2 shows that about 90 per cent. of the total breakage occurred within 10 in. (25.4 cm.) of the ends of the steel; in other words, in the portion of the steel heated for forging. Some of this breakage may be said to be caused by fatigue, but the existence of such a phenomenon in steel is doubtful and is denied by many investigators.

In order to get a better idea of the location of breaks at the ends of the steel, the curve shown in Fig. 1 has been plotted. This curve has been derived by plotting at the even inch, breaks that occurred within ½ in. (1.27 cm.) of that point. The position of maximum breakage in the tools is shown by the sketches of the shank and bit ends. The different

TABLE 2

Brand	Number of Tools	Breakage: Within 10 in. of Ends: Bit End	Breakage: Within 10 in. of Ends: Shank End	Breakage: Within 10 in. of Ends: Total	Breakage: Between Points 10 in. from Ends: Bit End	Breakage: Between Points 10 in. from Ends: Shank End	Breakage: Between Points 10 in. from Ends: Total	Total Breaks	Total Drilling Time: Hours	Total Drilling Time: Min.	Total Drilling Time: Sec.	Footage Drilled: Feet	Footage Drilled: Inches	Average Drilling Time per Break: Min.	Average Drilling Time per Break: Sec.	Average Footage Drilled per Break: Feet	Average Footage Drilled per Break: Inches	Average Inches per Minute
A..	4	4	0	4	0	0	0	4	3	54	30	129	0	58	37.5	32	3.0	6.60
B..	4	2	2	4	0	0	0	4	3	48	30	94	4	57	7.5	23	6.9	4.92
C..	7	13	7	20	3	0	3	23	15	24	10	560	4	40	10.6	24	4.3	7.26
D...	11	1	28	29	0	0	0	29	22	46	15	633	8	47	6.7	21	10.2	5.58
E.	24	3	50	53	3	16	19	72	26	10	35	636	0	21	48 8	8	10.0	4.86
F..	6	1	24	25	0	1	1	26	14	22	3	454	0	33	9.3	17	5.5	6.30
G.........	39	48	45	93	2	0	2	95	83	39	5	2758	4	52	49.9	29	0.4	6.60
Total	95	72	156	228	8	17	25	253	170	5	8	5265	8	40	20.2	20	9.7	6.18
Per cent. of total breaks.	..	..	...	90	..	..	10											

steps taken in forging and hardening the steel are given in order that the full value of this curve may be appreciated. The shank end is considered first. When a collar is necessary, which was not the case in the tests under discussion, the fresh steel is heated for about 6 or 7 in. (15.2 or 17.8 cm.) from the end and the collar is forged by upsetting. The condition of the metal at the farthest point of heating undergoes a rather sharp change from the annealed structure of the original material to a structure produced by the cooling from the forging temperature. The shank is then reheated for a distance, which probably varies up to 7 in. from the end, to a temperature above its critical range, but probably



IG. 1.—CURVE SHOWING BREAKS THAT OCCURRED WITHIN ½ IN. OF THE EVEN INCH. ALSO SKETCH OF SHANK AND BIT ENDS.

below the temperature for forging, and is plunged in oil. There is, then, a change from the annealed to the oil-quenched condition. When a lug or collar is forged there may, therefore, be a number of structurally different zones, depending on the length of the steel heated for hardening; namely, the original structure of the bar, that produced in the cooling of a part of the bar heated for the upsetting operation but not worked, that produced in the upsetting, and finally the oil-hardened structure of the end. When it is realized that, while the shank end of the steel is supported in the chuck of the drill, the shaft of the steel outside the drill is subjected to considerable vibratory and bending stresses, the stresses that this heterogeneous region is expected to withstand will be appreciated. It is no wonder that most of the failures from breakage are found in this region.

Much the same kind of operation is undergone by the bit end of the steel. About 4 in. (10 cm.) of it is heated for forging, and afterwards only ½ to 1 in. (1.27 to 2.5 cm.) is heated for quenching in water. Here, there are probably three zones of structurally different material,

TABLE 3

Brand	Number of Tools Tested	Number of Sharpenings	Total Drilling Time			Footage Drilled		Average Drilling Time per Sharpening		Average Footage per Sharpening		Average Inches per Minute
			Hours	Min.	Sec.	Feet	Inches	Min.	Sec.	Feet	Inches	
A	8	12	23	6	00	720	1	115	30.0	60	2.5	6.24
B	8	15	20	37	30	560	1	82	30.0	37	4.1	5.43
C	5	33	11	17	00	447	2	20	30 9	13	6.6	7.92
D	11	78	31	11	45	822	11	23	59.8	10	6.6	5.27
E	24	162	26	12	41	645	1	9	42.5	3	11.8	4.92
F	6	56	14	22	3	454	0	15	23.6	8	1.3	6.30
G	47	286	119	6	5	3782	4	24	59.2	13	2.7	6.32
Total	109	642	245	53	4	7431	8	22	59.0	11	6 9	6.04

and the breakage is maximum at about 3 in. (7.6 cm.) from the tip of the bit.

The conditions just outlined point to the necessity of closely controlling all the conditions for heating and cooling. This is borne out by the record of sharpenings in Table 3, which includes the sharpenings necessitated by breakage, because the records did not differentiate with respect to sharpenings necessitated by dullness and those made necessary as a result of breakage, the number of sharpenings having been kept and not the duration of the run for each. The small number of breaks encountered in brands A and B accounts, in a measure, for the great drilling time per sharpening. The total average drilling time per sharpening is practically 23 min. for an average footage of about 11½ ft. (3.5 m.), yet the maximum drilling time per sharpening for any tool was 6 hr. 43 min. and 30 sec. for a footage of about 245 ft. 9½ in. (61 m.). There is, therefore, room for considerable improvement of the average.

Conclusions

To summarize, drill steel should be in the best condition when received from the manufacturer, free from imperfections of a mechanical nature and from impurities; it should also be in the best annealed condition to withstand the vibration and shock its service will entail.

There is, for a given steel, a temperature and a rate of cooling from this temperature that will produce the conditions that will best withstand the stresses of shock and bending.

Lack of uniformity in performance shows a lack of uniformity in the heat-treating operations, which may be overcome by the devising of means, of an automatic or "fool-proof" nature, that will attain the desired result without calling for excessive skill on the part of the smith.

Experimentation with the heat treatment of the shank and bit ends of the steel might lead to alterations in the present method of heat treatment that would do away with or mitigate the evil effects of the sharply defined zones of structurally different material, which must result to a more or less extent from present practice.

Application of Magnetic Analysis to Rock Drills

By Charles W. Burrows, Ph. D., Grasmere, N. Y.

(New York Meeting, February, 1921)

The burden a man can endure depends on its magnitude and the number of times it is applied, as well as on many other factors. The resisting power of steel likewise is dependent on many factors. The maximum load it can support indefinitely is much less than its possible momentary burden. A light load straining the metal far within its elastic limit, if repeated indefinitely, ruptures the metal. The magnitude of the stresses that may be alternately applied and removed without rupturing the material depends on the number of repetitions of the cycle; the intervening periods of rest; the thermal, electric, and magnetic conditions, as well as the presence of secondary streessses. While we are unable to formulate the law between the magnitude of repeated stresses necessary to produce rupture and the length of time during which these alternating stresses may be applied before rupture, my experience is that the time increases as the magnitude of the force decreases, according to a law which, to a first approximation, is exponential. In other words,

$$F = Ce^{-t}$$

where F = magnitude of alternating stresses; t = time for which stresses are continued; C = constant, which depends on the conditions under which the operations are carried out. It is quite probable that the same type of law holds for the magnitude of a continuous force and its duration.

When a rod of steel is subjected to the blows of a drilling hammer several things happen. First, one end is struck by the hammer and the other end is driven into the rock, thus causing a bodily displacement of the entire material of the steel drill. The greatest stresses involved in this bodily displacement are located at the two ends of the steel and manifest themselves by the distortion of these parts. Second, after the blow has been delivered, the steel drill, due partly to its own resiliency and partly to the resiliency of the rock, experiences a vibratory motion as a whole, this motion is quite similar to the bouncing of a rubber ball thrown violently upon a smooth pavement. Third, the free vibration of the drill is determined only by its own natural period of vibration. This motion involves all the characteristics of any vibrating body emitting sound. Certain elements of the rod will experience alternations of condensation and rarefaction and, in these regions, the individual steel

fibers will be subjected alternately to stresses of tension and of compression. Certain other regions will be subjected to rapid displacements alternating on the two sides of a position of equilibrium. The individual fibers in these regions of oscillation are not subject to the great stresses of the fibers of the first class.

The physical theory of the rod vibrating after this manner is well known: (1) The rod may vibrate in such a manner as to produce its fundamental or lowest tone; in this case, the region of maximum stresses of tension and of compression lies at the middle of the rod. (2) The rod may vibrate in its first overtone; in this mode of motion, the rod vibrates in two halves with a frequency double that of its fundamental, the maximum stresses here occur at the middle of each of the vibrating halves. (3) The rod may vibrate in any number of segments. In any case the regions of maximum stresses of tension and compression lie in the middle of each vibrating segment.

The period of vibration is given by the quotient of the velocity of sound in steel divided by twice the length of the steel bar in question. In one case investigated, the velocity of sound in the steel is 17,000 ft. per sec. and the length of the bar is 3 ft. The corresponding period of vibration of the fundamental tone associated with this rod is approximately 3000 per sec. I did not succeed, however, in getting this rod to vibrate in its fundamental, although no difficulty was experienced in forcing the rod to vibrate in seven or more segments, vibrating with a frequency of 21,000 or more cycles per second. As the hammer delivered 30 blows per second, the regions of maximum stresses experienced 700 alternations of tension and compression for every blow of the hammer.

Cause of Breakage

A bar of steel in service may be considered as being made up of an infinite number of elements or fibers. Each fiber has its part to perform and rupture occurs when the individual fibers cannot withstand the stresses imposed upon them. This is a simple way of stating the case and yet, because of its simplicity, we often fail to understand the phenomenon.

If all the fibers of a bar of steel had the same resisting power and were subjected to stresses of the same kind of magnitude, we should expect the pieces of steel to fail throughout the entire mass at the same time, in the manner of the "one horse shay," and disintegrate into particles of dust. However, all steel fibers do not have the same mechanical properties and the stresses throughout the steel are not equal; consequently we may expect breakage to be localized over some region wherein the relation of the stress to the ability of the corresponding fiber to withstand this stress is greatest. Whenever stresses are strongly localized, the corresponding fibers are called upon to withstand a greater

burden and under long continued stresses of this nature will break. Again, stresses may be uniformly distributed throughout a mass of steel that contains certain localized elements of weakness, such as slag inclusions, blowholes, etc. In these regions we may expect a rupture to occur. Third, the structure may be so designed that in a given region the fibers must bear a more concentrated load than the other fibers; here again is a region of weakness. Finally, with careful design and good material, there may be initial thermal stresses that, under the action of the impressed stresses, produce a region of relative weakness where rupture may be expected.

Accelerated Tests

In an accelerated test, I employed an electric hammer kindly loaned by R. Morley of the Electro-Magnetic Tool Co. This hammer, as I used it, strikes against a small hardened steel piece having a shoulder

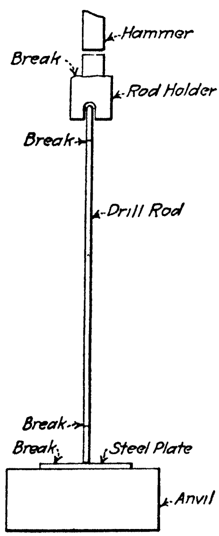


Fig. 1.—Relative positions of parts in accelerated drill test.

near its upper end and a cylindrical cavity for holding the end of the rod under test, which was a bar of tool steel ⅛ in. (3.17 mm.) in diameter and 3 ft. (0.9 m.) long. The other end of this rod rested upon a small plate of steel, which in turn was backed by a heavy anvil.

The hammer delivered light blows at the rate of 30 per second and within a few hours several distinct types of fracture occurred. The first break was at the shoulder of the hardened steel piece; this undoubtedly was due to the sharp angle at the shoulder which forced all the stresses

due to the repeated stresses to be strongly localized. As proof of this, a shoulder piece in which the sharp corner was replaced by a rounded fillet has already endured many times the service of the first and shows no signs of deterioration.

The second break occurred in the rod about one-fourteenth of its length from the ends. This break occurred near the end struck by the hammer and also near the end impinging upon the anvil. We would expect breaks to occur in these regions in accordance with the preceding theory if the bar vibrates in seven segments. The third break occurred in the small plate of iron lying between the drill rod and the anvil. This piece broke in a region of reduced cross section at some distance from the point where the blows were falling.

General Testing

Two conditions lead to the failure of a piece of steel: The nature of the steel may not be suited to the use to which the steel is put. This unsuitability of material may be due to improper composition, improper heat treatment, or other causes. Second, while the material may have the general characteristics required, it may be defective in certain regions due to blowholes, slag inclusions, or lack of uniformity in the heat treatment. These two conditions require two types of testing.

The usual method of testing steel and steel products consists in the examination of a limited number of representative pieces. The test is frequently destructive so that the material is no longer usable for commercial purposes; this is particularly true of the tensile test, in which the specimen is ruptured. The Brinell and scleroscope tests make small indentations on the test sample and thus mar or even destroy the finished surface. The chemical test is destructive of the entire portion tested. Many of the routine tests determine the characteristics of only a small portion of the sample tested.

In an effort to devise means of testing that do not have these objections, several other physical characteristics have been investigated in the hope that they may show sufficient differences to discriminate between the various grades of commercial material and of such a nature that they may be measured without excessive difficulty. Investigators have, therefore, subjected steel to the action of the X-ray; the above measured properties, in general, may thus be determined without destroying the test material. In certain cases, the physical differences measured in this way are of sufficient magnitude to discriminate between the usual grades of commercial material. An objection to the optical method is that it requires a special machined reflecting surface and, furthermore, it determines only the characteristics of the surface of the material. The examination of the effects of the X-ray, the electrical resistance, the thermal electromotive force, and the magnetic characteristics give values

that are a measure of the entire material under examination. The X-ray examination is limited to materials rather small in size. Electrical resistance calls for an elongated specimen. The thermal electromotive force requires that the specimen be heated to some extent. The optical and electrical tests are based on a single variable or, at most, only a limited number of variables.

The Magnetic Test

The magnetic test fulfils most of the requisites of an ideal test method. It is non-destructive of the test material, and may be applied to every piece of material in question. The complete magnetic examination of

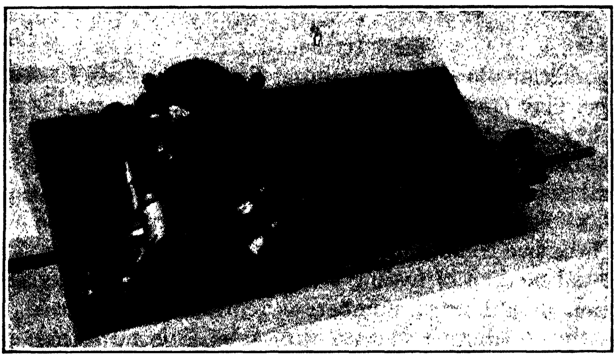


Fig. 2.—Defectoscope suitable for examination of drill rod. At the right are the rolls for driving the test material through the solenoid and back of the rolls is the driving motor. At the center of the magnetizing solenoid is a detector coil similar to those shown in Fig. 4.

a piece of drill steel is divided into two parts: the location of regions of weakness, where failure is liable to occur, and the estimation of the average nature of the material as resulting from its composition of heat treatment. The location of regions of weakness is determined by means of the magnetic defectoscope, shown in Fig. 2. The operation of this instrument is based on the fact that the amount of magnetic leakage from a bar in a magnetic field is strongly influenced by the presence of inhomogeneities in the steel.

We may recognize six elements in the magnetic defectoscope. First, the bar must be magnetized. In the general type of apparatus under discussion the magnetization is effected by a relatively short solenoid energized by a direct current of such value that the magnetization of the specimen is carried well beyond the knee to the induction curve. The second element is the means for detecting magnetic variations in the bar. This detector consists of two test coils having the same number of turns and surrounding the specimen. The magnetizing solenoid

and the detector are rigidly connected together and are given a relative motion along the length of the specimen by means of a suitable motor, which forms the third element of the system. As the detector occupies

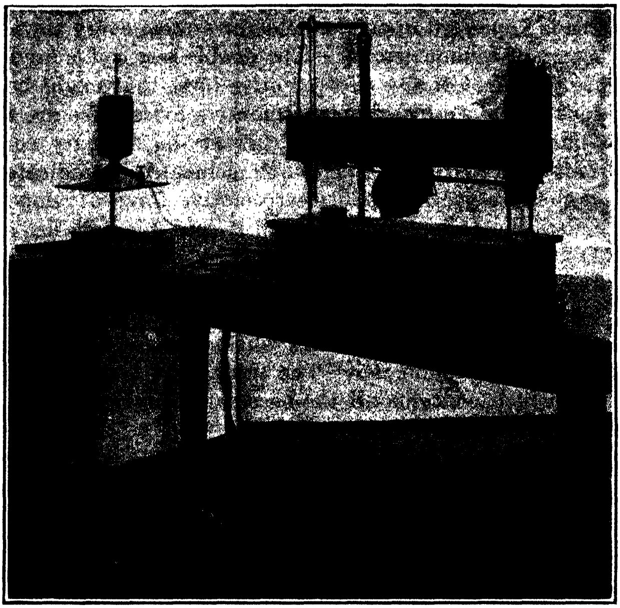


FIG. 3.—FRONT VIEW OF CAMERA, CONTROL BOX, AND GALVANOMETER FOR DEFECTOSCOPE. GALVANOMETER MUST BE OF MODERATE SENSITIVITY, FAIRLY RAPID PERIOD, AND OVER DAMPED. THE MOTOR BETWEEN CONTROL BOX AND CAMERA BOX DRIVES CAMERA FILM DOWNWARD AT RIGHT ANGLES TO SLIT THROUGH WHICH THE SPOT OF LIGHT FROM THE GALVANOMETER IS REFLECTED.

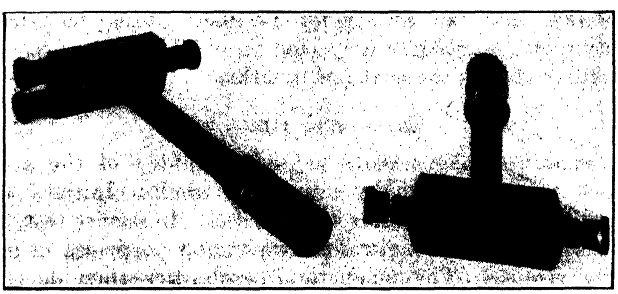


FIG. 4.—TEST COIL AND MOUNTING SUITABLE FOR A SMALL ROD OF DRILL STEEL.

different positions along the length of the test material, it is threaded by an induction, which depends on the nature of the corresponding region of the specimen. If the specimen is not quite uniform, the magnetic

induction threading one of the coils of the detector is different from the magnetic induction threading the other coil. The result is that the electromotive force generated in one of the test coils is different from that generated in the other. Consequently the small differential electromotive force is impressed upon the detector system every time it passes over the magnetic inhomogeneity. The double test coil is used in order to eliminate the effect of any electromotive force that might be induced by variations in the source of magnetomotive force. We are thus able to use as our source of magnetizing current the ordinary commercial direct-current power circuit. The fourth element is an indicator, which must be responsive to the small electromotive force developed in the detector coils. For this purpose we find it convenient to use a heavily damped D'Arsonval galvanometer of short period. The indication given by the galvanometer is recorded by the fifth element of the equipment. The recorder is essentially a photographic film caused to move uniformly across a small slit through whose opening a spot of light is reflected by the galvanometer. The sixth, and last, element is the control box, which contains all the necessary electrical switches, rheostats, and instruments.

The test determines the uniformity of the bar along its length. It does not, however, indicate whether the bar is uniformly hard or uniformly soft. In other words, a perfectly uniform bar of hardened steel will give the same indication for uniformity as a perfectly uniform bar of annealed steel. To determine the nature of the steel, we measure one or more of its specific magnetic properties in some type of magnetic permeameter. Investigations are now in progress to determine the specific magnetic characteristics that will give the most information concerning its mechanical characteristics and also the most desirable type of instrument with which to measure these magnetic characteristics. A committee of the American Society for Testing Material has this matter under active investigation and specific proposals may be expected from this committee during the next few months.

Magnetic Results

Rods examined magnetically at the beginning of the accelerated service test showed some very interesting results. Initially, the test rod shows only relatively minor irregularities. In service test, magnetic irregularities develop in regions corresponding to places of maximum alternating stresses. The magnetic irregularities thus developed increase in prominence as the test progresses. These modified regions are symmetrically arranged along the length of the rod and correspond to a vibration of the sixth overtone. In each of the two rods that broke under service, the break occurred in one of the regions indicated by the magnetic test.

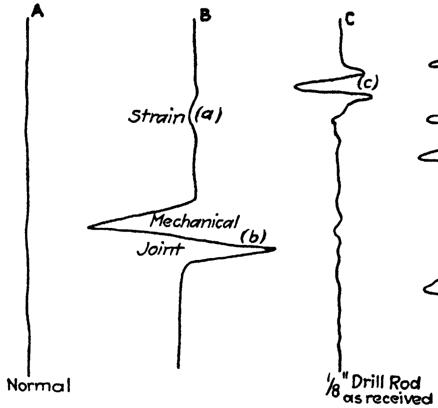


FIG. 5.—TYPICAL RECORDS OF DRILL-ROD STEEL. A. PERFECT SPECIMEN; B. STEEL HAS BEEN STRAINED AT (*a*) AND HAS A MECHANICALLY PERFECT THREADED JOINT AT (*b*); C. SHOWS A NORMAL HARD SPOT AT (*c*); D. STEEL CONTAINING MARKED IRREGULARITIES.

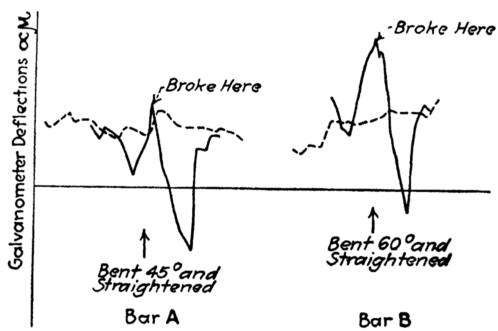


FIG. 6.—MAGNETIC EXAMINATION OF PIECE OF STEEL BEFORE AND AFTER BENDING; POINTS OF RUPTURE WHEN STEEL IS TESTED IN A TENSILE MACHINE ARE INDICATED.

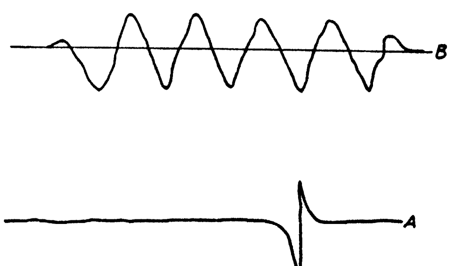


FIG. 7.—A. IDEAL CURVE SHOWING EFFECT OF A MECHANICAL DISCONTINUITY; B. ACTUAL CURVE SHOWING REGULAR STRAINS INDUCED IN STEEL RAIL THAT HAS BEEN RESTING ON SPECIAL TIES.

In Fig. 5, *A* is a typical magnetic record of a commercially perfect bar of steel. In *B* are shown two defects into a bar of otherwise perfect steel. At (*a*) is shown the result of bending steel through a sharp angle and restraightening. (*b*) is a typical effect observed when two pieces of steel are threaded end to end. *C* and *D* are the records of two steel drill rods. At (*c*) appears a marked irregularity due to a local hard spot in the rod. The other slight irregularities in *C* represent the normal variations that may be expected in the average piece of steel of fairly good quality. The variations of *D* are such as may be expected in drill steel of inferior grade.

In Fig. 6 are shown two rather interesting records taken on tool-steel bars both before and after they had been stressed beyond the elastic limit. The bars were finally broken in a tensile machine. In each case the fracture occurred in the region of marked magnetic inhomogeneity.

In Fig. 7, *A* is the characteristic form of record that indicates a complete discontinuity or a large flaw in a bar of steel. The record of the rail shown in *B* was taken some years ago and is probably the first complete evidence of the fact that strains are introduced into the rail due to the spacing of the ties in connection with service conditions.

Reclamation of Depreciated Steel

We are all familiar with the fact that steel experiences a structural fatigue after it has been called upon to resist repeated stresses for a given length of time. We believe that it is possible for steel to recuperate from such fatigue. There is some evidence that the mere freedom from applied stresses will permit the steel to recover, at least in part, its original strength. We have been told that when steel has been heated to a given temperature and subjected to the action of an electric current it becomes rejuvenated. We are also told that the simultaneous heating and magnetizing of steel tends to remove the effects of fatigue. It is my experience, which I believe is in accord with the experience of many others, that heat alone will remove the potential weakness resulting from continued service. Our knowledge on this subject is quite vague although the beneficial effects of heat treatment are generally supposed to require heating above the critical temperature. This heat treatment corresponds to annealing. My belief is that recovery from fatigue may be expected to result from cessation of alternating stresses for any length of time, with the material at any temperature. The degree of recovery is increased by extending the period of recuperation and increasing the temperature of the fatigued specimen. A quantitative experimental study of the degree of restoration and the conditions under which the restoration occurs would form a valuable contribution to technical science.

Résumé

While this investigation is far from being completed the results thus far obtained seem to indicate that the magnetic defectoscope will render a threefold service to the users and manufacturers of drill steel. It will indicate the existence of fatigue in the material; it will show the location of these fatigued regions; it will indicate whether efforts to reclaim the material have been successful.

Abstracts of Papers Presented in Drill Steel Sessions* New York Meeting, 1921

Investigation of Fatigue of Metals Under Stress

By H. F. Moore,† M.M.E., Urbana, Ill.

We have studied the simple case of repeated stress, without considering impact, which might bring in other factors. This investigation has shown that steel under repeated applications of stress, reversed from positive to negative, will not fail below some fairly clearly defined limiting stress that, so far we can see, does not bear any definite relation to the ordinary elastic limit, being as large as the elastic limit in some cases and about one-half the elastic limit in others.

According to one theory, fatigue is the result of changes in crystalline structure by repeated stress. I do not know of any evidence in favor of this theory. The crystals are broken under repeated stress. According to Bauschinger, under repeated stress some inherent property of the material changes its elastic limit; possibly some property of the amorphous cement between the crystals. A third theory is that all fatigue of metals is the result of the spread of damage from little localized overstresses. While I do not feel justified in accepting the last theory rather than the second, I have yet to run across a case of failure of steel or other metal, either in the laboratory or in service, that could not be explained by the gradual spread of damage from some nucleus.

We have found, for example, that it is possible to subject a homogeneous steel, like Armco iron or a 0.90 per cent. carbon steel, that has been thoroughly annealed, to one hundred million reversals of stress as high as the elastic limit of the steel with no sign of failure. On the other hand, a complex structured steel, like a chrome-nickel steel that has been hardened with very little drawing, will fail at a stress about 58 per cent. of its elastic limit. Between these two we get a variety of results.

In the steel work of any building, there are undoubtedly many thousands of places where structural damage was done—where riveted members were bent into place cold and where the rivets were fitted with drift pins, etc. Locally the steel in the building was stressed beyond its yield point in many places. These little localized overstresses do no damage at all; the building is entirely safe. But if this building were subjected eight or ten times a day to a mild earthquake, these little localized overstresses might be nuclei from which damage would spread

* Abstracted from *Mining and Metallurgy*, Nos. 174–176.

† Professor of Engineering Materials, University of Illinois.

and eventually we would have trouble. That is, these little localized overstresses may not be important if the structure is loaded a few times, but may be important if the structure is loaded many thousands of times.

The existence of localized stress has been demonstrated by direct experimentation with delicate extensometers by the elaborate investigations of Coker. Localized stresses four, five, and six times those that we would get by the ordinary laws of mechanics exist in most of our structures. These stresses, ordinarily, under steady load do no damage but if they are repeated many times may start progressive failure.

So far, we have seen no failure that could not be explained by localized high stress at a nick, a crack, a point where one heat treatment stopped and another began, or where there was local faulting, or any one of a dozen causes. In the case of drill rods, the actual distribution of stress must be different from that which any mathematical analysis could predict. It would seem that the spread of damage from the localized overstresses, so far as we can see now, explains the phenomenon known as fatigue.

The Ideal Drill Steel

By Frank H. Kingdon

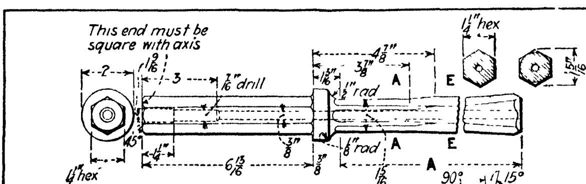
The essential qualities of a drill steel are: (1), It must be easily forged; (2), the forged bit end must permit easy heat treatment to obtain hardness to resist chipping; (3), the bar or body must be stiff to resist bending or twisting and yet tough to resist shock and vibration, with resulting breakage, (4), the forged shank end must be easily heat treated to obtain hardness with great toughness.

Minor requirements are that the bit and the shank are in alignment with the body; that the shank is of the proper shape and length and the shank collar or lugs of the proper diameter and length; that the hole is free from obstruction; that the striking end of the shank is flat and square; that the bit is of the proper shape, with the cutting and reaming edges formed full and to the required size; that the reaming edges are concentric with the axis of the steel, and that there are no sharp corners at the shoulder where the bit blends into the body.

The essential qualities of a bit, are: That its shape be such that maximum cutting speed can be maintained for as great a distance as possible before wear of the gage and cutting edge reduces the speed of penetration to a point where a change of steel is made necessary; that the size or diameter of the drill hole corresponding to the gage of the bit can be maintained with the least possible reduction as the depth of the hole increases, and that the bit can be correctly and readily formed and heat treated. The following features of bit design require attention: Shape,

total length, and angle of cutting edge; length and area of reaming edges or surfaces; size and shape of clearance grooves for ejection of cuttings, and length and angle of wings and manner in which they are blended into the body of the steel.

The longer the cutting edge the greater the amount of rock cuttings per blow, assuming that the cutting edge remains sharp enough for the conditions. The dulled or blunt edge, besides decreasing the penetration per blow, lessens the cushioning effect, causing the drill steel to rebound from the rock. In a radial cutting-edge bit the most work is done at the extreme cutting and reaming edges; therefore the only way to improve this condition is to shape the cutting edge so that the work is evenly distributed throughout its length and the extremities of the cutting edge have suitable reaming surfaces properly tapered back to improve the



Overall length of drill steel			2'-8"	4'-8"	6'-8"	8'-8"	10'-8"	12'-8"
Lengths tee will cut "A"			2'	4	6	8'	10'	12'
Size of bar stock			1 1/4"	1 1/4"	1 1/4"	1 1/4"	1 1/4"	1 1/4"
Dimensions at "D"			1 11/32"	1 5/16"	1 9/32"	1 1/4"	1 15/64"	1 13/64"
Dimensions at "N"			1 1/8"	1 3/32"	1 1/16"	1 1/32"	1"	31/32"
Dimensions at "C"			9/16"	9/16"	9/16"	9/16"	9/16"	9/16"
Dimensions at "R"			15/32"	7/16"	27/64"	13/32"	25/64"	3/8"
Gage of bit "B"	2 ft set	1	1 5/8"					
Gage " "	4 " "	2	1 11/16"	1 5/8"				
Gage " "	6 " "	3	1 3/4"	1 11/16"	1 5/8"			
Gage " "	8 " "	4	1 13/16"	1 3/4"	1 11/16"	1 5/8"		
Gage " "	10 " "	5	1 7/8"	1 13/16"	1 3/4	1 11/16"	1 5/8"	
Gage " "	12 " "	6	1 15/16"	1 7/8"	1 13/16"	1 3/4"	1 11/16"	1 5/8"

FIG. 1.—DOUBLE-ARC, DOUBLE-TAPER BIT.

wearing qualities of the gage. So far as I know the only bit that approaches the ideal bit is the double-arc, double-taper bit; Fig. 1 shows its characteristics.

The potentiometer system is the most reliable temperature-reading system for the drill-steel hardening shop. The furnace should be so constructed that it can be used for heating for forging and tempering. The burner should be placed so the flame does not hit directly on the metal; for continuous work the maximum heat zone should be at the end from which the drill steel is withdrawn; the pyrometer or thermocouple should be installed in this end.

In heating, a neutral or reducing atmosphere should be maintained to prevent scaling and decarburizing. The steel must be heated evenly and uniformly; too rapid heating may develop checks or cracks, and too slow heating may cause soaking, which tends to enlarge grain growth

For hardening, the steel should be heated just above the critical temperature, as a properly forged steel quenched from this heat has some toughness, with maximum hardness and density. The quenching bath should be kept at approximately a constant temperature. In drawing or tempering, longer time periods at lower temperatures are better than the shorter periods at higher temperatures.

A recent questionnaire brought out the following facts: The double-arc, double-taper bit and cross bit give the best service; resharpening of the drill steels most frequently was caused by wear of gage; chipping of bit was, with one exception, the second cause; breakage of shank was with one exception, the third cause; upsetting of bit, upsetting of shank, and breakage of body were the fourth cause. The direct cause of the necessity of resharpening given was poor heat treatment first, and faulty steel and severe rock conditions second.

Rock-drill Steel

By N. B. Hoffman

Much of the drill steel produced in America is manufactured into hollow rods. After all forging has been finished the entire bar should be thoroughly annealed and heat treated before the point is hardened and tempered, at least where the drill has heavy duty, such as working in deep holes and hard rock.

A large number of tools of like character and doing work requiring similar properties are now being made of alloy steels; for instance a steel with a carbon content of 0.75 to 0.85 per cent., manganese 0.25 to 0.35, phosphorus under 0.020, sulfur under 0.030 and vanadium 0.18 to 0.23 has proved very satisfactory for chisels. A steel of this type properly manufactured will be far superior to the ordinary carbon steels, while at the same time is one whose price range should not prove prohibitive. Much of the cutting efficiency of a drill depends on proper heat treatment. This vanadium steel has a far greater safe hardening and forging range than ordinary carbon steels, making it more nearly foolproof. It will harden nicely at 1400° F. but will also harden well and satisfactorily at 1550° F. retaining its quality of hardness and toughness in either case.

Drill Steel from Hollow Ingots

By P. A. E. Armstrong

Hollow drill steel is made from a drilled billet with a sand-filled core or the drilled, pierced, or the drilled and pierced billet, not sand-filled, is rolled down over a projectile or ball much the same as in ordinary pipe manufacture. The first is the method generally used in this country; the second is used largely in Sweden. Swedish hollow drill steel has a

world-wide reputation for excellency, but steels made by other methods are efficient.

In the Swedish material the hole is badly decarburized by the method of manufacture; the extent of this decarburization varies in bars of different makes. Nothing that can be discerned microscopically or by analysis proves this steel to be superior to hollow drill steel of good manufacture made by methods that prevent decarburization of the hole. Hardened bars with this decarburized hole do not become so intensely hard on the inside, neither are they so prone to crack. The mild steel or decarburized iron inside the hole acts as a damping effect when hardening is produced. The thicker the decarburized zone, the less will be the hardness. To harden right across the cross-sections alloy steels, which have a greater lag to transformation, must be used.

Hollow drill steel made by casting the metal around a tube and mechanically working the hollow ingot down to the required size (filling the hole with sand and later removing it after rolling) will have an artificially produced equivalent decarburized core which will be free from any such tendency toward splitting and driving in radial cracks. The surface of the inside of the tube will be comparatively smooth, giving a uniform cooling rate to the inside of the bar, again reducing the tendency for a radial crack to start.

Segregation must occur in all steel solidification and is responsible for a certain mechanical weakness in the finished bar. The segregation will be concentric with the hole and its maximum occurrence is about midway between the exterior of the ingot and the tube wall. As this area is large in circumference, comparatively speaking, the segregation is comparatively small because the distance between the tube and the ingot mold is small. This thin layer of segregation will be most noticeable at the top of the ingot. It can have, at most, little effect on the bar, as the segregates do not, under any circumstance, creep through to either the exterior or the interior wall of the ingot. Segregation is not removed in billets pierced from soiled ingots and is only indifferently removed in drilled billets. Even if segregated impurities are removed by using only the butts of the ingots, and drilling the billets, there is still carbide segregation to contend with. This will run in straight lines parallel with the wall of the ingot or the line of freezing. The inside of the hole of a drilled billet must, therefore, necessarily be the weakest place in the billet, resulting in a weak bar. Progressive fracture is easily started from one of these many sources of weakness.

In a general way the smaller the ingot, the better the steel. In a small ingot, the crystals are not large, the crystallites not far-reaching, and the speed of cooling fairly rapid. Crystallites are also apt to have a uniform direction for some distance, depending on the temperature and the speed of cooling. If the steel is cast hot and quickly enough, a

fractured ingot will show a complete diagonal structure. If the temperature is lowered, the equi-axed zone is larger and the pine tree growth is less. A small ingot, though, is prone to this columnar structure, even though the casting temperature is kept down.

The speed of cooling after solidification is slower in the hollow ingot than in the solid, therefore the granulation, which takes place after the crystals have formed from the liquid metal, is apt to produce larger grains than in the solid ingot. This is not a detriment. The original crystal size is of much more importance. The size of the grains due to this granulation constantly varies in response to mechanical work and subsequent heat treatment.

Surface weakness, as shown by seaming and checking on the exterior when working or by cracking in hardening, is in some measure the result of the surface stresses of the ingot when cast. The temperature of the ingot mold or speed of freezing of the exterior of the ingot is of great importance and much seamy and cracked steel is caused by a too quickly cooled skin. Ingots that are turned or milled all over will not seam very much in working, in fact they are practically free from seams, whereas an ingot rolled without a preliminary machining operation is full of seams and cracks, some long and some short.

It has been said that these short seams and cracks in the surface of the bar are the result of elongated depressions in the surface of the ingot and that if an ingot is cast very smooth it should give fewer surface defects. This is partly true, but an ingot cast very hot and having a very smooth skin will crack and seam very badly in the mill, as will the surface of an ingot poured very cold, which is very crinkly, because of the low temperature of the metal. The cause of the cracks and seams in one is not the cause of the cracks and seams in the other.

The tube method of hollow drill steel manufacture is superior to the older method because of: (1) Greater freedom from external and internal straining; (2) the inherent small crystal size; (3) absence of harmful segregation, resulting in weakness of wall of hole. (4) less liability for steel to crack in inside of hole during forging or hardening; (5) toughening effect, arising from mild steel wall of hole limiting the intense hardening on quenching.

Breakage and Heat Treatment of Rock-drill Steel

By A. E. Perkins, Pittsburgh, Pa.

Actual figures showing the superiority of one section or another are difficult to obtain. In a given size of steel, however, the cruciform section shows the most breakage; it is inherently a weak section. It contains less steel per foot than any other standard section of the same size and when rolled the wings tend to cool off setting up cooling stresses.

The best steel for a fixed set of conditions is the steel that will withstand the duty imposed upon it without undue breakage and will withstand the abrasive action of the rock with the least amount of wear. Low-carbon steel will stand more abuse by the blacksmith and will lend itself more readily to welding than steel embodying a greater percentage of carbon. Under the average conditions the following specifications are right for straight carbon drill steel:

	Per Cent.		Per Cent.
Carbon	0.85 to 0.90	Phosphorus	0.03
Manganese	0.30 to 0.40	Sulfur	0.03

In buying drill steel one must not rely too much on chemical analysis for the quality of a steel that must be hardened and tempered depends far more on the process of manufacture and the character of the workman than on the chemist.

The points that make for maximum service from drill steel may be summarized as follows:

Never heat either the bit end or the shank end, preparatory to forging, higher than 1600° F. Ninety-five per cent. of the trouble underground is the result of overheating in forging or in tempering or both.

Always leave the steel in the furnace long enough to insure its being heated uniformly and thoroughly; but steels lying too long in the fire, especially when not turned, are liable to show soft spots in hardening.

Quench at the critical temperature on the rising heat.

When tempering shanks heat treat for 2 in. below the lugs or collar. Density and toughness provide wear-resisting qualities, while hardness, except on the extreme outer surface, is not essential or desirable.

Either keep the heated tool moving about in the quenching bath or keep the bath agitated.

Watch the drill-sharpening machine for worn dies and dollies.

Watch the air pressure at the sharpening machine; most sharpeners should have at least 80 lb. at the machine.

Keep the chuck bushing and the drill shank up to size.

Drill-steel Sharpening

By Clarence M. Haight, Franklin, N. J.

The general practice in drill sharpening shops, of which descriptions have been written, is about as follows: The bit is heated to 1600° to 1900° F. Then when forged to the proper shape and size in the sharpening machine, the steel is allowed to cool; it is then passed to a hardening furnace where only the end is heated to about 1450° F., and then quenched in cold water, brine, or in brine with oil on top.

If the assumption is true that steel retains the grain size that corresponds to the highest temperature to which it has been subjected, there

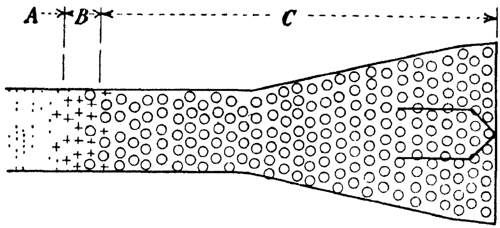


Fig. 1.—Steel as taken from the fire. A, normal size; B, section of change from normal size to 1800° F. size; C, size at 1800° F., 4 in. long.

is a condition in the bit and shank that, perhaps, is detrimental for the best service from the steel.

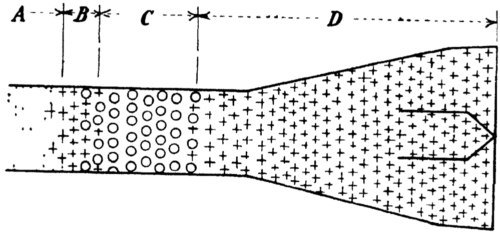


Fig. 2.—Steel after forging. A, B and C as in Fig. 1. D, size of grain at 1600° F., 3 in. long from hammering; this leaves C only 1 in. long with a section of change between C and D.

For the sake of clearness let the following assumptions be made: That the forging temperature is 1800° F. but that the forging operation

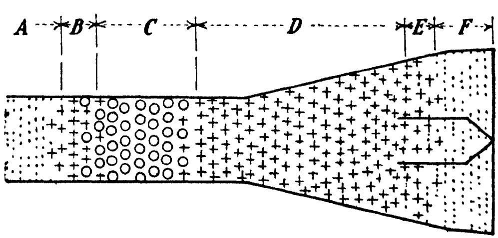


Fig. 3.—Steel after hardening. A, B, C and D as in Fig. 2. E, section of change from 1600° F. grain size to normal grain size; F, normal grain size in hardened portion; D has been reduced in length by E and F.

reduces the grain size to a size corresponding to that produced at 1600°; the grain size desired is that produced at 1450°.

If the bit is originally heated for 4 in. back from the cutting edge,

the hammering effect acts only to a point 3 in. back from the cutting edge, and the hardening heat extends back only 1 in. from the cutting edge. Then, as a steel is taken from the fire, the grain condition is as shown in Fig. 1. After the forging operation has been performed the condition shown by Fig. 2 exists. The heating operation to harden the cutting edge produces the effect shown in Fig. 3.

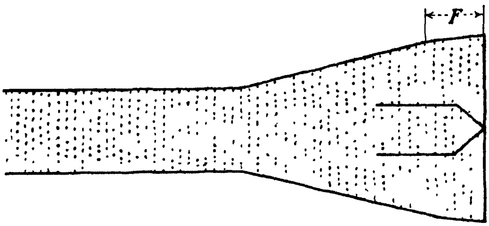


FIG. 4.— CONDITION DESIRED WHEN STEEL IS READY FOR USE.

The questions that naturally arise are: Will the cost of the extra heat treatment necessary to change this condition (annealing) be more than the saving that may be effected in the drill steel or will the extra heating be an economical as well as a wise thing to do? Is this condition necessarily detrimental to the maximum service of the drill steel? Can this annealing action be obtained in the hardening heat by heating more of the bit, say at least 5 in., to the hardening temperature but quenching only the end of the bit by a shallow immersion?

SHORT ROCK-DRILL STEELS RECLAIMED BY WELDING

BY W. T. OBER, LYNN, MASS.

Experimental tests by the Thompson Company have shown that short lengths of rock-drill steels may be reclaimed by electric butt welding. The drills used in the experiments were hollow drills 1¼-in. in diameter containing 0.95 per cent. to 1.05 per cent. carbon.

The ends to be joined must first be ground off fairly square and the greater part of the rust, scale, etc. removed for 3 or 4 in. back in order to insure good electrical contact with the dies of the welder and countersunk, with an 80° countersink, approximately ¼ in. The steels are then clamped in the copper dies of the welder, so that a length projects from the inside edge of each pair of dies equivalent to from one-half to three-quarters the diameter of the drill. The drills should also be brought into accurate alignment.

The first step is to bring the ends into close proximity and then close the primary circuit of the transformer, slowly bringing the steels together until their ends lightly touch, when they will "flash," that is, minute

particles of molten steel will fly off. The circuit should be kept closed and the ends of the steels slowly forced together until the ends are "flashed" off square and are in even contact around the entire periphery. Usually, if the ends are properly prepared, a forward movement of about 1/16 in. of right-hand steel will accomplish this. Do not "flash" too much or the water hole will become clogged.

At the end of the flashing movement a quick forward movement will bring the ends of steels into solid contact and stop the flashing. Now, with little or no added pressure on steels, allow the joint to reach the welding temperature, when instantaneous and heavy pressure should be applied and the steels forced together, completing the weld.

A flux of plain powdered borax is sometimes used but it is not absolutely necessary, though it is recommended.

The question of removing the "upset" at the weld is one for the user to decide; the greater the amount of this upset that can be retained, the stronger the bar at that point. Every weld should be properly heat treated to relieve any locked-in stresses and to refine the grain structure.

BIOGRAPHICAL NOTICES

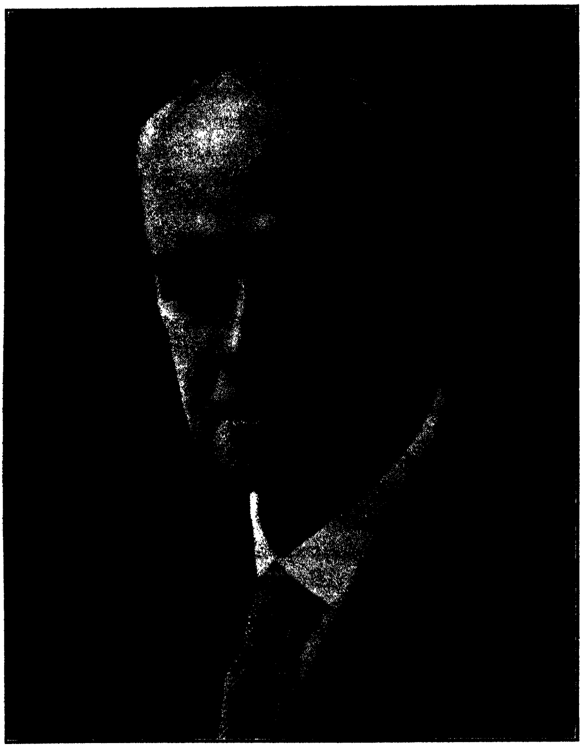
Hennen Jennings

HENNEN JENNINGS was born at Hawesville, Ky., in 1854. His father had settled in that place in 1853, having gone thither with the idea that the coal resources of the central states were destined to be of great industrial importance and having made up his mind to participate in their development. This family association with an important branch of the mining industry without any doubt was an influential factor in determining the career of the son who was born in the following year. Another influential factor was the friendship existing between the Jennings family and the Smith family, the latter living on the other side of the Ohio river in Indiana, and being also interested in coal mining. Hamilton Smith, the son of the latter family, was a good many years older than Hennen Jennings, but the latter as a boy knew him well, looked up to him, and contemplated following in his footsteps. Thus, it was but natural that Hennen Jennings after completing his preliminary education was attracted to mining and went to the Lawrence Scientific School at Harvard University. There he met Prof. N. S. Shaler, the distinguished geologist, who was a potent influence in shaping the careers of many American mining engineers at this time. Professor Shaler had had practical experience in the development of mineral resources and had acquired an outlook far broader than the merely academic, and, moreover, was gifted with the ability to impart this, together with his own enthusiasm, to the students who came under him.

Hennen Jennings was graduated in civil engineering from the Lawrence Scientific School in 1877. He proceeded immediately to California to take a position on the staff of the North Bloomfield mine, a famous hydraulic mine of that time, and was employed at the Bowman dam. The direct inspiration for this venture was his previous acquaintance with Hamilton Smith, who was then the president of the North Bloomfield company, and its consulting engineer. H. C. Perkins was the general manager and Hennen Jennings became his assistant. Thus began a friendship that was life-long. I may pause here to remark that all of the friendships of Hennen Jennings were life-long and it was largely owing to his quality of inspiring such friendships that his success in life was so distinguished.

Having remained for some time at the North Bloomfield mine, Hennen Jennings left to go to the New Almaden, as assistant to Ross E. Browne, who was then the surveyor of that property. After some years at the New Almaden he returned to the North Bloomfield as assistant manager, under H. C. Perkins. Leaving that position he went to the Ruby mine

in Sierra county, a channel gravel deposit whereof he instituted the development, his friends Smith and Perkins being associated with him as stockholders in the company. After a while Mr. Smith made up his mind that the outlook for the property was not good and that it would best be abandoned. Mr. Jennings, however, having perfect confidence in his own judgment, even against that of Mr. Smith, bought Mr. Smith's shares and continued the exploitation of the deposit until it was worked



HENNEN JENNINGS.

out, achieving a brilliant success not only for his reputation but also materially, this being the first mining operation in which he acquired substantial financial returns for himself.

The Ruby mine having been worked out, Hennen Jennings went back to the New Almaden, this time as superintendent. It was at about this time that he met Miss Mary Coleman, daughter of J. C. Coleman, owner and manager of the Idaho mine at Grass Valley. He married her in 1886, and she survives him, together with two children. His last con-

nection with the New Almaden mine was not very long, for in 1887 he was summoned to take charge, as manager, of El Callao mine, in Venezuela. El Callao was a famous gold mine that had been turned down by the Rothschilds upon the advice of Hamilton Smith, who regarded the price asked for it as being unreasonable, but the Venezuelan owners were so much impressed by Mr. Smith that they asked him to select a manager for their property and he first sent H. C. Perkins. Mr. Perkins being obliged to leave it, Hennen Jennings was dispatched thither as his successor. El Callao was famous not only for itself, but also as being the nest whence the American engineers flew when they were called upon to institute development of the gold mines of the Witwatersrand. From El Callao went H. C. Perkins, Hennen Jennings, George E. Webber, Barry Searles, Louis Seymour, A. M. Robeson, and George Poore. It was owing to Hamilton Smith having become interested in the Witwatersrand, in behalf of his clients, that he summoned the members of his old staff.

Hennen Jennings first went to South Africa as consulting engineer for the group of mines controlled by H. Eckstein, with which house he remained for 10 years. While in that capacity he displayed the qualities that had previously distinguished him but were now enhanced by his riper experience. These qualities were fundamentally his loyalty to the men working under him, and reciprocally and naturally their loyalty to him; his hospitality to new ideas; and his sound judgment with respect to whatever he was called upon to do. Hennen Jennings was distinctly a well-balanced man, one who was able judicially to weigh evidence and never was in any danger of being carried away by wild ideas, or being misled by will-o'-the-wisps.

Soon after Hennen Jennings arrived on the Rand, James A. MacArthur and J. H. Cordner-James came out with the MacArthur-Forrest cyanide process and built a plant for its exploitation at the Salisbury mine. Hennen Jennings became interested in this, had visions of its possibilities, and with his assistants spent days and nights at the Salisbury mine, studying the operation of the process. Having made up his mind that it was destined to be a metallurgical and commercial success he advised his principals to go into it, which report led to the erection of the first big cyanide plant on the Witwatersrand, this being at the Robinson mine.

Another very important idea that Mr. Jennings quickly assimilated was that of picking out by hand the waste rock that was inevitably broken with the ore in mining the thin veins of the Rand. This was first suggested by Harry John of the Ferreira mine. Mr. Jennings quickly perceived the merit of the idea and put it into practice on a large scale.

Either of these innovations would have been sufficient to establish his reputation, even if he had done nothing else, but the genius of Mr.

Jennings indicated greater things. He was fortunate in possessing an analytical mind and had a strong tendency toward economics. These faculties enabled him to see things in their broad perspective and led him to direct his attention to the major problems. He could leave safely the details of administration to his colonels and captains who were bound to him by the magnificent feelings of loyalty that he inspired. This left him free, as the general, to plan and execute the big campaigns. One of the first things that he saw, once he had familiarized himself with the conditions of the Witwatersrand, was the economic advisability of enlarging the scale of operations. At that time the 20-stamp mill was about the standard of practice. Hennen Jennings became convinced that these ore deposits were of unparalleled magnitude and extent and were capable of exploitation upon the biggest scale that the boldest imagination indicated, and he possessed such an imagination. He was one of the first, moreover, to grasp the idea that a mine is a wasting asset and that its present value, anyway in the case of a gold mine, is increased by speeding up the rapidity of exhaustion. In this he anticipated the principles that were subsequently formulated so lucidly by Hoover.

Mr. Jennings had a hard time and many long fights in convincing his principals of the soundness of his views, but eventually he won, and he was able to carry out his plans for mining and milling on a gradually increasing scale, so that by the time he left the Witwatersrand the standard of milling operations had risen to about 240 stamps per unit.

While in South Africa Mr. Jennings by no means confined himself to the narrow field of his own work. Then, as subsequently in his distinguished career, he was animated by the desire to do general good and he gave freely of himself to public affairs. He was greatly interested in the subject of technical education and believed that every country should supply its own technical men. He foresaw the time when the American engineers would leave the Witwatersrand, their services being no longer required and their places being taken by men developed locally. He assisted generously in such development. He sat on two commissions that considered this problem, and he and Theodore Reunert were mostly responsible for the establishment of the South African School of Mines, at Johannesburg, about 1903. Mr. Jennings was also greatly interested in the formation of the South African Institution of Engineers, and was its first president.

Hennen Jennings remained with H. Eckstein & Co. until 1897, being consulting engineer for the Robinson, Ferreira, Crown Reef, New Heriot, City & Suburban, Village Main Reef, New Modderfontein and other mines. In 1897 he went to London, becoming consulting engineer for Wernher, Beit & Co. He went out to South Africa again in 1902, returning to London in 1903. While living in London, during this period, he was honored by election, in 1903, to the presidency of the Institution of

Mining and Metallurgy, and was the only American ever thus honored. In 1904 the gold medal of the Institution was awarded to him "in recognition of his services in the development of the mining industry in South Africa, in improving the status of the mining and metallurgical professions, and in the cause of mining and metallurgical education in London and South Africa." Maintaining his interest in technical education, he was appointed a member of the departmental committee, appointed by the Government, that studied the Royal School of Mines, to consider in what manner the staff buildings and appliances might be utilized to the fullest extent for the promotion of higher scientific studies and to report on any changes necessary to carry out the recommendations that should be made. The report of this committee was adopted and greatly increased the scope and usefulness of that ancient and honorable institution.

Hennen Jennings retired from his connection with Wernher, Beit & Co. in 1905 and in 1906 returned to America, establishing his home in Washington. While in South Africa he had added substantially to the fortune that he had started while still a young man in California, and in taking up his residence once more in his native country he retired from active practice. He did not, however, cease all activities, for he became consulting engineer for the Conrey Placer Mining Co., operating gold dredges at Ruby, Mont., in which he and some of his friends were interested; but the chief owner was the Gordon McKay Estate and eventually Harvard University. Mr. Jennings took up the work of directing this company largely from his desire to help his friend, Professor Shaler, and his alma mater. Now and then he accepted some professional commission such as that of membership on the committee which a few years ago reported upon a plan for the consolidation of the Treadwell group of mines. In the main, however, he confined himself to public affairs. He accepted the chairmanship of the Division of Mining, Metallurgy, Economic Geology and Applied Chemistry of the Second Pan-American Scientific Congress, which sat in Washington, Dec. 27, 1915, to Jan. 8, 1916, and contributed to that congress a scholarly paper on the "History and Development of Gold Dredging in Montana, with a chapter on Placer Mining Methods and Operating Costs," which was subsequently published as a bulletin by the U. S. Bureau of Mines. He also accepted an appointment as consulting engineer in the U. S. Bureau of Mines. With the latter appointment he was made chairman of the committee selected by Secretary Lane to study the gold situation, and in that capacity he prepared and published a report that is a classic on the subject. Ever since his return to America in 1906 he had been thinking deeply about it and probably there was no other man in the United States capable of expressing such authoritative judgment. In this paper he exhibited powerfully the same tendencies toward economic

thought that had distinguished his career as a consulting engineer in South Africa.

Mr. Jennings maintained an unflagging interest in Harvard University and liberally endowed a scholarship there, the income from which was to be given to a student in engineering of ability, who had difficulty in paying his way through college. In 1918 his alma mater conferred upon him the honorary degree of master of arts, with the citation "Hennen Jennings, an eminent consulting engineer, whose advice is sought in matters of great importance from San Francisco to London and from London to Johannesburg."

During the war Mr. Jennings was a member of the Platinum Committee of the War Industries Board and was also associated with other committees that functioned in Washington. He was a member of all of the professional societies and his professional colleagues loved to honor him and would have signalized him by any preferment that he would have been willing to accept. His modesty, however, forbade his accepting much that his professional colleagues would have enjoyed doing. There was perhaps no man in the entire profession who was more honored in the hearts of those who knew him and those who knew of him.

With respect to his friendships I have already spoken. That with H. C. Perkins lasted for more than 40 years, Mr. Perkins also having returned to America and also having established his home in Washington. A few days ago Mr. Perkins wrote the following:

> After an intimate acquaintance and friendship with Hennen Jennings for over 40 years I wish to speak enthusiastically of him as one of the strongest men intellectually and morally whom I have known. He was well grounded in the fundamental principles of his profession, and his methods in his work were sound and thorough In his relations with others he was most kindly, humane and human. By his death the world is made poorer, and his friends have lost a very lovable companion.

During the last two years Mr. Jennings' health, previously robust, failed and for 14 months he was ill, but during this period of waning strength his clear and forceful mind never lost its power and continued to dominate, with patience, courage, and a firm hold upon the work that he hoped to resume. The end came without warning and without pain, March 5, 1920.

W. R. INGALLS.

Irving A. Stearns

IRVING ARIEL STEARNS died at his home, 60 South River St., Wilkes-Barre, Pa., on Tuesday morning, Oct. 5, 1920, of pneumonia, after an illness of about a month.

In his death, the Wyoming Valley loses one of its most eminent and best loved citizens, and the loss of so strong a participant in the activities of the Valley will leave a gap most difficult to fill. Despite his more than three score and fifteen years, Mr. Stearns, at the time of his death, was president of the Wilkes-Barre City Hospital, Wyoming National Bank, and the Wyoming Historical and Geological Society, chairman of the Board of the Vulcan Iron Works, and a director of the Spring Brook Water Supply Co., and of the Wilkes-Barre Lace Manufacturing Co.

Mr. Stearns came of New England stock, a descendant in the eighth generation of Charles Stearns who was admitted freeman in Watertown in 1646. Irving Ariel Stearns was born Sept. 12, 1845, in Rushville, N. Y., a son of George W. and Miranda (Tufts) Stearns. He was graduated from the Rensselaer Polytechnic Institute of Troy, N. Y., in the celebrated Class of 1868.

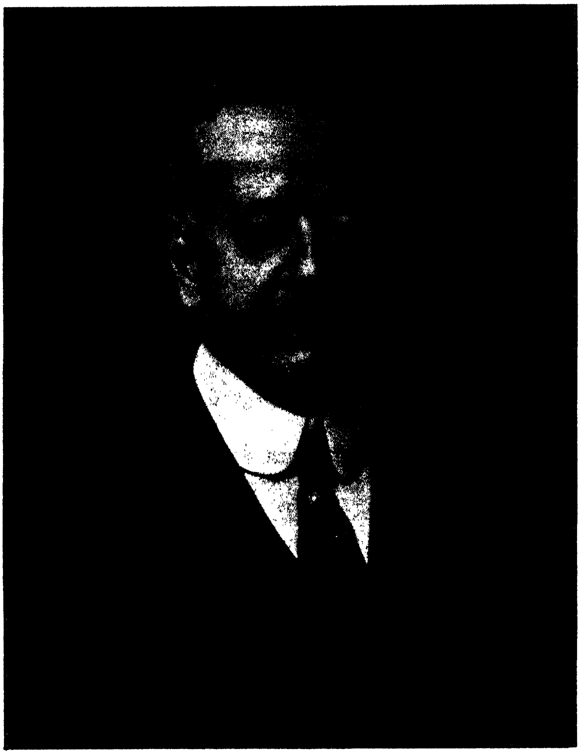
After graduation, he was assistant professor of chemistry in charge of the analytical laboratory for a year, then went to Wilkes-Barre in 1869, as engineer in the office of R. P. Rothwell, at that time the leading mining engineer of the region. From 1871 to 1872, he was superintendent of the McNeal Coal & Iron Co. of Schuylkill County, Pa. and in 1872 succeeded to Mr. Rothwell's business when the latter became editor of the *Engineering & Mining Journal* of New York.

From 1872 to 1885, Mr. Stearns was in private practice as a mining engineer, handling a great amount of business in the anthracite region, including the building of the bridges at Shickshinny and Pittston, and the surveying and mapping of many of the individually operated mines. His practice extended throughout the country, with numerous examinations and reports upon mining properties in Pennsylvania, Virginia, West Virginia, Arkansas, Colorado, California, Wyoming, Idaho and Utah; he was responsible for the design and execution of the Lehigh Valley Railroad Co.'s great Tifft Farm improvements at Buffalo, N. Y. including canals, docks, coal stocking plant, etc.

He was commissioned Quartermaster of the Ninth Regiment, N. G. P., March 29, 1880, was promoted to Major on May 15, 1884, and resigned Apr. 1, 1885. He always retained his interest in the National Guard and was trustee and president of the Ninth Regiment Armory Association.

At an early date, Mr. Stearns was one of the best known mining engineers in the United States, and his eminence in his profession led to his appointment, in the fall of 1885, as manager of the various coal interests owned and controlled by the Pennsylvania Railroad Co., which position he held until July, 1897.

During this time he brought the properties of which he was in control to the highest state of efficiency, he was personally responsible for the



IRVING A. STEARNS.

introduction in the anthracite region of high-pressure boilers, the first of which were put in at Shamokin; of underground electric haulage, the first in the United States and the second in the world, put in the Lykens Valley colliery in 1886; and of high-pressure compressed-air haulage in 1895. He also introduced radical improvements in mining and in the preparation of anthracite coal.

In July, 1897, Mr. Stearns was elected president of the Cross Creek Coal Co., Coxe Brothers & Co., Inc., the Delaware, Susquehanna &

Schuylkill R. R. Co., and the Coxe Iron Manufacturing Co., and held these positions until the sale of the properties to the Lehigh Valley Coal Co. in the fall of 1905. During this time, with his genius for organization, he brought the mines to the highest degree of operating efficiency.

The sale of the Coxe properties found Mr. Stearns sixty years of age and tired from thirty-seven years of continuous and most active service, so he decided to retire from business and devote the remaining years of his life to broader channels of usefulness.

As soon as he was free from direct business engagements, his abilities were heavily drawn on by this community. On Nov. 30, 1906, he was elected the first president of the Wilkes-Barre Park Commission, and held this office for ten years. At that time the only park lands owned by the City of Wilkes-Barre were two tracts patented in 1804, one known as the River Common extending from South to Union Streets along the Susquehanna River, and the other the Public Square. During his two terms as Park Commissioner, the River Common was changed from an eyesore to its present condition, the Public Square was remodeled, Hollenback Park, Riverside Park, Frances Slocum playground, and numerous small parks and playgrounds throughout the city were acquired and improved, giving to the city its present park system, acquired almost entirely by gift as the result of Mr. Stearns' personal efforts.

Mr. Stearns' clear-headed judgment and sound common sense were in great demand by the business interests not only in the Wyoming Valley but throughout the country. He was at different times director of the Lehigh Valley Railroad, Lehigh Valley Coal Co., Chatham & Phoenix National Bank of New York, Standard Trust Co. of New York, the Hibbard-Rodman-Ely Safe Co., of New York, Spring Brook Water Supply Co., Spring Brook Water Co., Wilkes-Barre Water Co., Wyoming National Bank, Vulcan Iron Works, the Gas Co. of Luzerne County, Wilkes-Barre Electric Light Co., Peoples Telephone Co., and the Penn Mining Co. of Wyoming, of which latter he was president.

Mr. Stearns was a member of many societies and clubs; he was one of the organizers of the American Institute of Mining Engineers, founded in Wilkes-Barre in 1871, was vice-president in 1905–06, and at the time of his death was one of the four living original members of the Institute now on its rolls. He was the first president of the Westmoreland Club and an active member throughout its entire history, a past-president of the Wyoming Valley Country Club, and at different times a member of the American Society of Civil Engineers, the Union League Clubs of New York and Philadelphia, the Engineers and the University Clubs of New York, the Franklin Institute of Philadelphia, and of the Blooming Grove Club of Pike County, Pa. He took great interest in the Wyoming Historical and Geological Society, of which he was president for many years, and in the Wilkes-Barre City Hospital, of which he was on the

board for more than a generation and of which he was president at the time of his death.

Mr. Stearns married, Nov. 20, 1872, Clorinda W. Shoemaker, the eldest daughter of Hon. L. D. and Esther (Wadhams) Shoemaker of Wilkes-Barre, who died May 6, 1904. Of their three children, Captain Lazarus Denison Stearns, born Dec. 27, 1875, gave his life to his country as Captain in the Ninth Regiment N. G. P., during the Spanish-American War, on Sept. 6, 1898; Irving Ariel Stearns, Jr., born July 5, 1877, died Apr. 9, 1884, of scarlet fever; Esther Shoemaker Stearns married, Apr. 14, 1910, Harold Mercer Shoemaker and she and their two children, Irving Stearns Shoemaker and William Mercer Shoemaker, are the only surviving descendants. Mr. Stearn's only sister, Mrs. A. J. Aldrich, lives at Coldwater, Mich.

Mr. Stearns was a life-long attendant of the First Presbyterian Church, and was active in the building of the present edifice. He was a man of rare judgment, sound common sense, a good sportsman, a good friend, and a good citizen.

R. V. Norris.

T. Wada

TSUNASHIRO WADA, honorary member of the Institute, died at his residence, Ushigome, in Tokyo, on Dec. 20, 1920, at the age of sixty-four; he was born on March 15, 1856, at Obana in the province of Wakasa. An expert mineralogist, lithologist and geologist, he identified himself throughout his life with the development of the mining interest of this country. Perhaps the most noteworthy of his meritorious services in the public sense was the extension of the Imperial Steel Works at Yawata, anticipating the grim fate that hurled Japan into a war with Russia in 1904.

Educated in the Imperial University of Tokyo by foreign professors, he studied especially mineralogy, lithology and geology. In 1875 he was appointed assistant professor in the Imperial University of Tokyo, and in 1878 entered the service of the Home Department.

From 1880 until 1893, Mr. Wada was Director of the Imperial Geological Survey of Japan, during which time he practically laid the foundation of the geological investigation work in Japan. In 1884, he made a trip to Europe by government order to attend the international Congress in Berlin, visiting en route several countries of Europe and America. While Director of the Survey, he was also, from 1880 to 1885, lecturer, and from 1885 to 1891, professor of mineralogy and lithology in the Imperial University, and, from 1889 to 1893, was Director of the Imperial Mining Bureau of Japan. In this capacity, he revised the mining laws of Japan, thus laying the foundation of Japan's mining administration.

In 1894, he was again appointed lecturer in the Imperial University of Tokyo; in 1895, appointed Technical Adviser on Mines to the Imperial Household Department of Japan; and, in 1897, appointed President of the Imperial Steel Works, which was completed in 1900 with the expenditure of $10,000,000. In this capacity he did a great deal to promote the iron and steel industry of this country. We owe him the ore-purchase contract with the Daiya iron mine in China, which remarkably facilitated the extension of the iron works in later days. The victory of the Russo-Japanese war must be said to be due to his merit in no small measure. He resigned the presidency of the Imperial Steel Works in 1901.

In 1899, Mr. Wada visited America and several countries of Europe by government order; in 1902, he traveled through north and middle China doing exploration work and studying the geology of the country, accompanied by mining engineers and geologists.

From 1896 to 1908, Mr. Wada was president and honorary member of

the Japanese Institute of Mining Engineers (Nippon Kogyokai), and in 1905, by the special requirement of the late Prince Ito, was appointed General Adviser of Mining to the Korean Government, which position he continued to keep at the request of General Terauchi after the death of Prince Ito, even after the annexation of the Peninsula to Japan.

Mr. Wada organized the Association of the Principal Mine Owners of Japan (Kozan Konwa-Kai) in 1908, and presided. As a representative of this organization, he was a noteworthy member of the committee that welcomed the Institute party on its visit to Japan in 1911.

In 1916, he formed the Metallurgical Research Institute (Kinzoku-Kogyo Kenkyu-sho), and presided. In the same year, he purchased the Cottrell patent from the International Precipitation Co., Los Angeles, with a view to its application to smelters and factories in Japan, the results of which have testified to the unerring accuracy of his foresight.

In 1917, Mr. Wada was nominated by his Majesty as a member of the House of Peers, in recognition of his past meritorious career; he was practically the first and the only mining expert in the House.

On several occasions, he was decorated by His Majesty the Emperor of Japan and the governments of different nations. On report of his critical condition, he was promoted by one rank to the 4th rank of the Senior Grade and two orders from the 4th to the 2nd Order of Merit.

Mr. Wada published "Mineralogy" in 1876; "Crystallography" and "The Minerals of Japan" in 1877; "The Mining Industry of Japan," in English, in 1893; and, in 1904, revised and published "The Minerals of Japan " in both Japanese and English with numerous photographs of crystals. After that he continually published "Beitrage zur Mineralogie von Japan" written in English and German as supplements for the former.

In five great national exhibitions of Japan, the first of which was held in 1877 and the last in 1901, he was always appointed a director as well as juror in the mining and metallurgical section.

The life of this remarkable man may be said "to begin and end" with the progress of the mining industry and the development of geological investigation in Japan; his whole efforts in public life were concentrated in this single field.

Mr. Wada is survived by his wife, Madam Saki; four sons, Mikio, Goro, Rokuro and Shichiro; and three daughters, Mrs. S. Sakuma, T. Hasegawa and Mrs. I. Goto. M. OTAGAWA.

At the annual meeting of the Board of Directors of the American Institute of Mining and Metallurgical Engineers, held Feb. 15, 1921, the following resolutions were adopted;

RESOLVED, that the American Institute of Mining and Metallurgical Engineers has learned with the deepest regret of the death, on Dec. 20, 1920, of Dr. Tsunashiro Wada, Honorary Member of this Institute, and desires by this Minute to give expression to the sentiments aroused by this event.

Be it therefore resolved:

That the American Institute of Mining Engineers acknowledge the eminent services rendered by Doctor Wada to the cause of science by his researches in his native country, Japan, as well as in China and Korea, and by his numerous publications on the geology and mineralogy of these lands;

That we recognize the important work done by Doctor Wada as Director of the Imperial Geological Survey of Japan, as Director of the Imperial Mining Bureau of Japan, as Technical Adviser in Mining to the Imperial Household Department, as President of the Japanese Institution of Mining Engineers, and as Professor of Mineralogy and Lithology in the Imperial University at Tokyo;

That we fully and heartily appreciate the value of his work as a contributor to a good understanding between Japan and the United States, as well as between Japan and all other nations, and trust that by the coöperation of scientific workers and leaders of thought, both here and in Japan, all apparent dissensions will be removed, and the cause of peace and progress furthered and extended;

That a copy of these resolutions be sent to the family of the departed scientist and to the various great institutions in Japan which he honored and made the greater by his wonderful personality and charm, and to all these the expression of our earnest sympathy in this bereavement which is international and in which we all share.

Edmund Gybbon Spilsbury

EDMUND GYBBON SPILSBURY, mining and metallurgical engineer of international reputation, died suddenly of heart failure on May 28, 1920, in the New York Eye and Ear Infirmary, following an operation for cataract, which had been successfully performed a few days before.

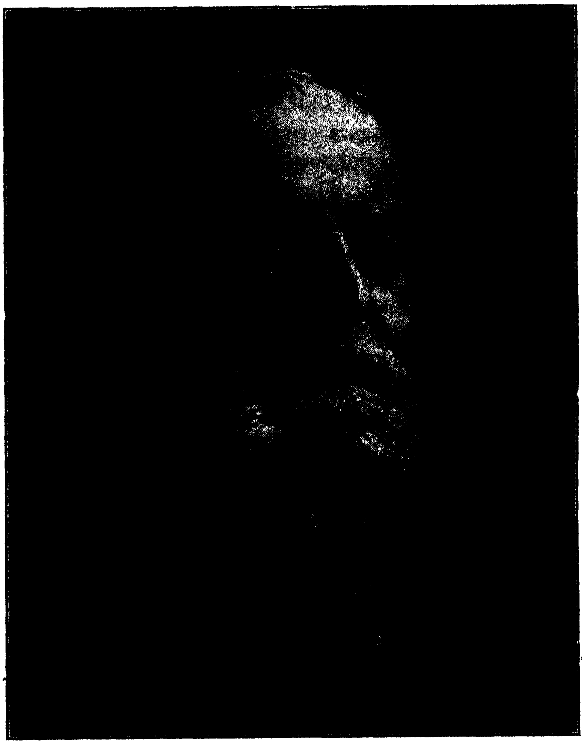
Mr. Spilsbury was one of the early members of the American Institute of Mining Engineers, which he joined in 1873. During all this time he contributed freely of his time and ability. He was manager of the Institute from 1885 to 1887, vice-president, 1893 and 1894, and president in 1896. He was also a member of the Engineers Club, of which he was president in 1916 and 1917, American Society of Civil Engineers, American Society of Mechanical Engineers, Institution of Mining and Metallurgy, Mining and Metallurgical Society of America, American Electrochemical Society and the Rocky Mountain Club of New York. He was a trustee of the United Engineering Society from 1916 to his death; a member of United Engineering Society Library Board from its organization in 1913 until his death, being its chairman from 1918 to January, 1920; a trustee of Engineering Foundation Board from 1916 until his death; and member of the John Fritz Medal Board of Award. He was also a member of the Division of Engineering of the National Research Council.

Born in London, England, in 1845, Mr. Spilsbury went to Liége, Belgium, at an early age, where he received his preliminary education. For his technical education he attended the University of Louvain, Belgium, graduating in 1862, and later he took a practical course at Clausthal, Germany.

In 1864 he became assistant engineer for the Eschweiler Zinc Co., of Stolberg, one of the largest miners and smelters of lead and zinc in the world at that time, and the next year he took charge of that company's mines and works on the Island of Sardinia. From Sardinia he went to the Atlas Mountains in Morocco. In 1867 he entered the service of McClean & Stilman, of London, and had charge of the construction of the iron gates for the Surrey Commercial Docks. In 1868 he was designing engineer with J. Casper Harkort, and had charge of most of the detail work of the Keulenberg Bridge in Holland, the Danube Bridge in Vienna, and the Rhine Bridge at Düsseldorf.

In 1870 Mr. Spilsbury came to the United States to investigate the lead and zinc resources for the Austro-Belgian Metallurgical Co. After spending two years in this work, he resigned in order to practice in the United States, and he was the first to introduce the Harz system of ore

dressing for the zinc ores of Pennsylvania and New Jersey. From 1873 to 1875 he was engaged as general manager of the Bamford (Pa.) Smelting Works. In 1879 he designed and built the Lynchburg Blast Furnace & Iron Works, and was also consulting engineer for the Coleraine Coal & Iron Co., of Philadelphia. In 1883 he became general manager of the Haile gold mine in South Carolina, and in 1887 engaged with Cooper Hewitt & Co., of New York. From 1888 to 1897, he was managing



EDMUND GYBBON SPILSBURY.

director of the Trenton Iron Co., Trenton, N. J., during which time he introduced the Elliot locked wire rope and the Bleichert system of aerial tramways.

In 1893 Mr. Spilsbury presided over the sessions of the mining division of the International Engineering Congress at the World's Fair, Chicago.

He was the author of a number of important technical papers in the Transactions of the American Institute of Mining Engineers, among

which may be mentioned: "On Rock-Drilling Machinery" (1874); "New Air Compressor" (1879); "Iron Ore Deposits on the James River" (1880); "Gold Mining in South Carolina" (1883); "Chlorination of Gold-bearing Sulfides" (1887); "Notes on the General Treatment of the Southern Gold Ores and Experiments in Matting Iron Sulfides" (1887); "Notes on a Novel Cable Transfer for Railroad Cars and the Use of the Locked Wire Rope" (1891); "Improvements in Mining and Metallurgical Appliances During the Last Decade" (1897) and "Improvement in Cyanide Process" (1910).

In lighter vein he wrote entertainingly. To the *Mining and Scientific Press* he contributed in 1915 "Technical Reminiscences," covering his interesting experiences during a half century of active practice.

Mr. Spilsbury's practice as a consulting mining engineer and metallurgist took him into many parts of Europe, Africa, the United States, Mexico and South America. During the winter and early spring of 1920, he spent a number of weeks in Brazil on a mining project for clients in the United States and had returned to New York only a few weeks before his decease.

Mr. Spilsbury is survived by three sons, Raymond G., Persifor G. and Hugh G.; by one daughter, Miss Beulah G.; and a sister, Miss Matilda Spilsbury.

Benjamin Bowden Lawrence

THE passing of Benjamin Bowden Lawrence in January, 1921, was a distinct loss to the engineering profession. Mr. Lawrence had a genius for reviving abandoned mines and developing them into substantial producers. A notable example of his work in this line was the Dives-Pelican near Georgetown, Colo. While on an exploring expedition in 1888, he examined this old mine and was convinced that bonanza ore still existed there. He located the owners of the property, and induced them to place sufficient capital at his disposal to enable him to re-open and explore this mine. So confident was he of the outcome that he undertook this work on a profit-sharing basis without salary.

Within eighteen months the old Dives-Pelican was operating profitably, and for the following ten years it paid very substantial dividends. The late Theodore N. Vail was one of the owners of this old mine, and many years later, at a dinner given to Mr. Lawrence by the Alumni of Columbia University, Mr. Vail told the story of the resuscitation of the Dives-Pelican by Mr. Lawrence, and declared that the dividends he himself had received provided a large part of the funds that made possible the realization of his dream of establishing the long-distance telephone in this country.

The El Cobre mines at Santiago de Cuba also owed their renewed life to Mr. Lawrence. Under his management, these mines produced and shipped several million pounds of copper per year. He was for several years consulting engineer for the Kerr Lake and other silver mines in the Cobalt region; the Kerr Lake became famous as a silver producer.

Mr. Lawrence spent more than twenty years of his life in the mining camps of the West. He was but twenty-one years of age, and only a few months out of college, when he was appointed manager of the great Chautauqua Lode (a silver mine) located on Glacier Mountain near Montezuma, Colo. Here, isolated in the mountains for several months each year by ice and snow, he worked for five years to make these mines pay.

He possessed unusual breadth of vision and human understanding; he had "personality plus," and when he opened an office in New York City as consulting engineer he quickly rose to an enviable position in his profession. Young engineers worked with enthusiasm under his guidance. He carefully selected his men, discussed fully with them the plans and policies to be pursued, then left them free to accomplish results in their own way. This attitude developed individuality and resourceful-

ness in his men, and their affection for him and their loyalty to him were good to see.

Mr. Lawrence was an enthusiastic mining man, and his great interest always lay in the real production of the metals. He wanted nothing to do with stock-jobbing in mining shares, and quickly severed his professional connection with any mining company that followed such practices. He enjoyed the enviable reputation of stating his conclusions without fear or favor, regarding any property upon which he was asked to make a report.

At the time of his death, Mr. Lawrence was president of the Cuba Copper Co., Phoenix Mines Co. and the Utah Metal and Tunnel Co. He was a director in the Low Moor Iron Co., the Washington Water Power Co. and the Research Corporation. He was consulting engineer for the Horn Silver Mining Co., the Utah Southern Mining Co., and the Cornucopia Mines Co. of Oregon. His clubs were the Down Town Association, University, Columbia University and the Union Club; his golf clubs were Oakland and Apawamis.

Mr. Lawrence was a graduate of Columbia School of Mines, and received from his Alma Mater the honorary degree of Master of Science. He was one of the presidents of the Alumni Association of the School of Mines and the first trustee elected to represent that body on the Board of Trustees of Columbia University. Shortly after the expiration of his term, a vacancy having occurred, he was elected a life Trustee and was made chairman of one of its most important committees—the Committee on Education. He was an enthusiastic member of the American Institute of Mining and Metallurgical Engineers; also of the Mining and Metallurgical Society of America.

No memoir of Benjamin Lawrence would be complete without mention of his interest in young men and his helpfulness towards them, particularly those in whom he thought he detected good timber for the engineering profession. His office latchstring was always out for such as these, and the number he helped through the course in mining, and the number he placed in their first engineering jobs after graduation, will never be known, these being matters which he considered of interest only to himself and the youths benefited.

Benjamin Lawrence was a man of enthusiasms; he got large returns out of life which he shared with others. A visit with him broadened one's horizon and sent one from his presence mentally swinging across the hilltops. He was very human; a staunch friend, and too big to feel enmity towards anyone. The place he occupied in the hearts of his friends and in the engineering profession will long be vacant.

At the Annual Meeting of the Board of Directors of the American Institute of Mining and Metallurgical Engineers, on Feb. 15, 1921, the following Minute and Resolution were unanimously adopted:

By the death of Benjamin Bowden Lawrence on Jan. 21, 1921, the mining profession lost one of its most eminent and high-minded engineers, and this Institute one of its most valued and beloved members.

Member of the Institute since 1882; Manager, 1903–1904; Member of the Council, 1905; Vice-president, 1910–1911, he ever brought to the service of the Institute the zeal and infectious enthusiasm which was one of his most compelling attributes, and which with his high ideals of professional conduct made his influence widely felt in many distant fields of mining activity as well as among his colleagues and friends at home, but especially among the younger members of the profession and students of engineering, who always found in him a sympathetic counselor, a practical helper and an inspiring example.

These qualities were accompanied by a lovable personal character, loyal and generous, exhibited in every relation of life.

In this expression of sorrow at his removal from earthly activity, and gratitude for his honorable and fruitful life, we echo the utterances of a world-wide circle of business and professional associates and loving friends.

RESOLVED: that the foregoing Minute be entered upon the records of the Board and that a copy thereof be sent to the family of Mr. Lawrence with assurances of deep sympathy in their bereavement.

William B. Cogswell

William B. Cogswell, member of the Institute since 1872, died on June 7, 1921, at his home in New York City, after an illness of about six weeks occasioned by an infection of the middle ear. Mr. Cogswell is survived by his wife, Mrs. Cora Brownell Cogswell; a son-in-law, and three grandchildren. His first wife died in 1877.

William Cogswell was born at Oswego, Sept. 22, 1834, and was taken to Syracuse by his parents when he was four years old. He attended Hamilton Academy and private schools in Syracuse and Seneca Falls. In 1843, he visited England with his parents and brought back a set of drawing instruments, which led to a series of lessons in architecture under Luther Gifford, the only architect in Syracuse at the time. The boy drew many building plans and one set was used for the old Globe Hotel in Syracuse, erected in 1846 and 1847.

In 1848, when only 14, he spent a year with a party that was surveying the route of the Syracuse & Oswego R. R. and relaying the tracks of the Utica & Syracuse R. R.

He studied civil engineering at the Rensselaer Polytechnic Institute, Troy, for three years, but left that institution in 1852 without receiving his degree. Many years later, in 1884, the degree of C. E. was conferred upon him.

During the three years following 1852, Mr. Cogswell served as an apprentice in machine shops at Lawrence, Mass., under the superintendence of John S. Hoadley. About 1856, he was made manager of the machinery department of the Marietta & Cincinnati R. R., at Chillicothe, Ohio, and three years later became superintendent of the Broadway foundry at St. Louis, Mo.

In 1860, in Syracuse, Mr. Cogswell was associated with William A. and A. Avery Sweet in the formation of Sweet Bros. & Co., which later became the Whitman & Barnes Mfg. Co.

At the outbreak of the Civil War, Mr. Cogswell received a civilian appointment as mechanical engineer in the United States Navy. During 1860 he was located at Port Royal, S. C., having general supervision over the work of fitting up repair shops at five widely separated stations on the Atlantic seaboard and Gulf of Mexico. In this year his genius and resourcefulness were notably shown. He literally launched a naval machine shop, so that warships could be repaired without leaving, and thereby weakening, blockades against Southern ports. The machine shop was assembled by Mr. Cogswell and shipped by boat to Port Royal. There an old whaler was made over for machine shop purposes and Mr.

Cogswell became its captain. One example of the efficiency of this floating machine shop is that a cylinder head, weighing more than 500 lb., was cast and made ready for a monitor.

In 1862, Mr. Cogswell was transferred to the Brooklyn Navy Yard to take charge of all machine repairs and mechanical construction. He continued that work until 1866, and spent the two succeeding years in New York City.

In 1869, Mr. Cogswell was called to Oneida County to supervise the construction and operation of the blast furnaces of the Franklin Iron Works, and at about the same time he was given charge of the work of completing the Clifton suspension bridge at Niagara Falls, which occupied him until 1873.

In 1874, when he was 40 years of age, he accepted an offer to take charge of the extensive lead mines controlled by Rowland Hazard at Mine La Motte, Mo. Five years of mining experience brought a new tendency and a bent for things under the earth as well as on land and water. Then came the meeting of the man and the big idea which was to mean so much to thousands. The man and the idea came together at a meeting of the American Institute of Mining Engineers, at Baltimore in February, 1879, where Mr. Cogswell heard Oswald J. Heinrich, mining engineer of Drifton, Pa., read a paper on the manufacture of ammonia soda.

The account of the bringing of the Solvay process to America, given by Mr. Cogswell, during the course of birthday reminiscences in 1916, was substantially as follows:

> The more I thought of Heinrich's paper on the manufacture of ammonia soda, the surer I felt that Onondaga County, with its limestone belt and salt and facilities for obtaining coal, furnished the field and the opportunity for a like endeavor.
>
> Not having had a vacation in five years, I went to Europe in the fall of 1879. Deciding to investigate the ammonia soda proposition, I obtained a letter of introduction to Ernest and Alfred Solvay of Brussels and stopped to see them while on my way from London to Hanover. They listened but refused the suggestion that a plant be established in the United States for the manufacture of soda ash under the process which they had developed into a commercial success.
>
> Later I wrote from Hanover, again asking the Solvays to consider the matter. They replied by requesting that I furnish letters of recommendation. After about three months—I was still in Hanover—there came a letter saying that the recommendations were very satisfactory and that if I had business in Brussels they—the brothers—would be glad to have me call on them.
>
> The outcome of the second visit to Brussels was that in 1880 I returned home with a commission to examine eligible sites for a plant in this country. I fixed on Syracuse as the most suitable location, reported back to the Solvays and they were satisfied.

The Solvay Process Co. was organized in 1881 and the work of erecting the original plant, now almost lost in the midst of the great plant of today, was begun. Rowland Hazard was made president, Earl B. Alvord,

William A. Sweet and George E. Dana, directors, and Mr. Cogswell, treasurer and general manager. When, in 1887, a change in the list of officers took place, Frederick R. Hazard, son of the first president, was elected treasurer and Mr. Cogswell was made vice-president and managing director.

The daily output of the original Solvay Process plant was 30 tons of soda ash. To-day this company is the largest manufacturer in the United States of soda ash (carbonate of soda) and its derivatives, and furnishes a larger part of all the alkali consumed in this country.

Mr. Cogswell's mining experience and geological research were of inestimable advantage in the successful search for deposits of rock salt for the Solvay plant. He worked upon the theory that the Syracuse salt springs were somewhere near the edge of a bed or vein of rock salt. Finally, after serious opposition, in 1888, 22 miles south of Syracuse near the village of Tully, the belief of the geologist was established. Rock salt was found at a depth of 1200 ft. This vein was found to be from 50 to 100 ft. in thickness, and beyond a vein of equal thickness was found.

To convey this salt in brine to the Solvay Process Works, Mr. Cogswell tapped one of the Tully lakes and brought the water through a pipe by gravity, discharging it into half a hundred wells, the solution then being piped to the Solvay plant by the Tully pipe line. In 1889 the Tully Pipe Line Co. was incorporated.

The mechanical engineer showed forth again and again in Mr. Cogswell's plans. Until the Jamesville quarries of the Solvay Process Co. were opened in 1911, limestone was transported from Split Rock to the works by a cable basket line, which was an interesting feature in the landscape for several miles.

Other notable work by Mr. Cogswell included his part in the development of the Hannawa Falls Power Co. at Hannawa Falls, St. Lawrence County, and at Colton in the same county.

Although Mr. Cogswell's charities were many, few persons knew about them. One of his greatest benefactions in Syracuse was the Hospital of the Good Shepherd, which he rebuilt and to which he made many gifts.

When he saw that some young man had "something in him," Mr. Cogswell sent him to college, and no one except the benefactor and the beneficiary knew about it.

Mr. Cogswell was a member of many scientific societies and of various clubs. Of the four Founder Societies, he belonged to three: A. I. M. E., A. S. C. E., and A. S. M. E. He was interested in architecture and astronomy, and had a remarkable collection of precious stones.

Stuart M. Buck

STUART MANWARING BUCK, a member of the Institute from the first year of its existence, and one of its Managers from 1883 to 1885, died at his home in Bramwell, West Va., on July 16, 1921, at the age of 79 years. He was the son of David and Matilda S. Buck of Boston, and

STUART M. BUCK.

was educated at Williams College, at the Massachusetts Institute of Technology and at the School of Mines at Freiberg, Germany.

Mr. Buck was one of the pioneers in the development of the coal fields of West Virginia. From 1872 to 1877 he was connected with the Kanawha and Ohio Coal Co. as engineer and as lessor; from 1877 to 1888 he was general manager of the East Bank Coal and Coke Co.; from 1888

to 1900, president and general manager, Norfolk Coal and Coke Co.; from 1900 to 1902, general manager, Dry Fork Coal Land Co.; from 1902 to 1904, president and general manager, Sagamore Coal and Coke Co. In 1904, he began the consulting practice which he continued until his last illness. He was also a director in the Pocahontas Fuel Company.

Mr. Buck was married in 1872 to Grace Ross, of Bangor, Maine, who survives him, together with three children, Clifford R. Buck of Philadelphia, Mrs. Edward C. Sherman of Washington, and Miss Theda Buck of Bramwell, W. Va.

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[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics.]

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